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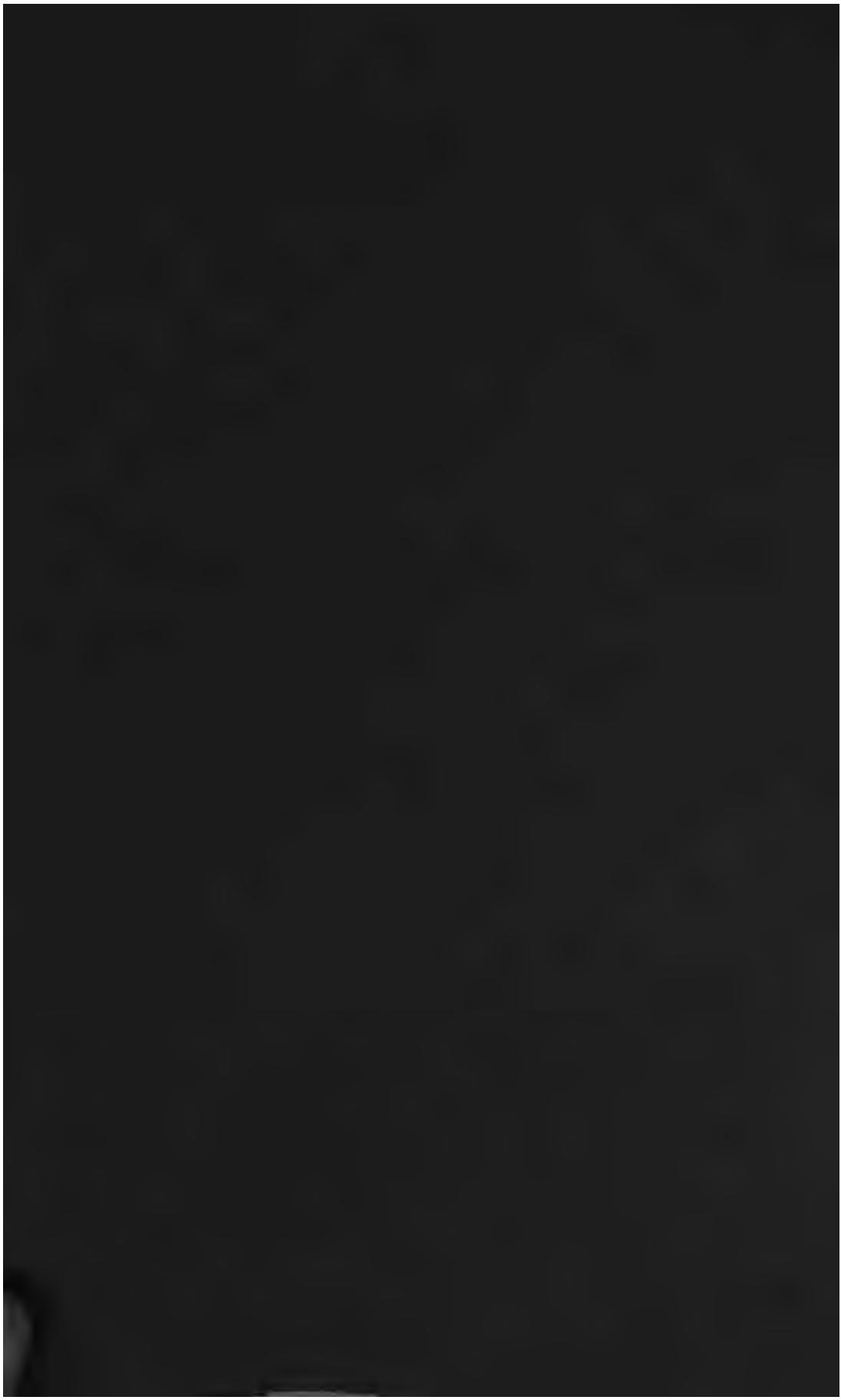

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METALLURGY

OF

SILVER, GOLD, AND MERCURY

IN THE

UNITED STATES.

THE

METALLURGY

OF

SILVER, GOLD, AND MERCURY

IN THE

UNITED STATES.

BY

THOMAS EGLESTON, LL.D.,

PROFESSOR OF MINERALOGY AND METALLURGY, SCHOOL OF MINES, COLUMBIA COLLEGE.

IN TWO VOLUMES.

VOL. II.

GOLD AND MERCURY.

JOHN WILEY & SONS,

53 EAST TENTH ST., NEW YORK,

AND

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1890.

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TO

LOUIS GRUNER,

INSPECTOR-GENERAL OF MINES
OF FRANCE:

A LEARNED PROFESSOR,

A GREAT INVESTIGATOR,

AND

A DISTINGUISHED SCHOLAR

THESE VOLUMES ARE GRATEFULLY INSCRIBED

BY

HIS PUPIL,

THE AUTH(

PREFACE.

THESE two volumes are the results of notes taken during very extended tours made in the years 1874 1887, 1888, and 1889, in which all the principal metallurgical establishments of the United States and its Territories were visited and carefully studied, and of constant correspondence with former pupils and others. The plan of the work has been to present the general principles of the treatment, and then afterwards examples of the best practice under each one of the general heads.

The publication of this volume has been retarded by circumstances entirely beyond my control, but it is hoped that the two volumes will present a succinct and clear statement of the metallurgy of silver, gold, and mercury in the United States in such detail as it is possible for so rapidly growing and changing a science to be presented in volumes of this kind. The treatment of silver bearing lead ores requires a separate volume.

In addition to the persons to whom I have acknowledged assistance in the first volume, I wish to express my thanks and obligations to Mr. W. McDERMOTT, of this city; to Messrs. RIBBON and MARCH, of Jersey City; to Mr. J. W. HENDY, to Mr. R. B. NOYES, and to Mr. L. A. PELTON, of San Francisco; to Mr. J. P. LAMBLING, of Ione, Cal.; to Mr. KNIGHT, of Sutter Creek, Cal.;

to Mr. N. W. Crocker, of the Bunker Hill Mine, Amador City, Cal.; to Mr. W. S. Davis, of Mamouth Bar, Cal.; to Colonel F. von Leicht, of New Almaden, Cal.; to Mr. G. H. Robinson, of the Drumlummon Mill, Marysville, Montana; and to Mr. A. Thies, of the Phœnix Mine, N. Carolina. These gentlemen have given me a great deal of information, and have helped me to carry out the plan of presenting in compact form the principles and practice of the silver, gold, and mercury works in the United States.

School of Mines,
New York, *May 1st*, 1890.

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THE METALLURGY

OF

SILVER, GOLD, AND MERCURY

IN THE

UNITED STATES.

CHAPTER I.

SHALLOW PLACER DEPOSITS.

ALTHOUGH it is less than forty years since gold was discovered in California, it is already very difficult to ascertain what the truth of the history is. This is owing to the fact that long before the real discovery, gold which was found in Mexico was credited to California, because it was brought to the East by vessels which reported it as coming from there. It is also owing in part to the fact that but little was known of the country, and that travellers who wrote about it, in the desire to feed the appetite for the marvellous, reported what they imagined rather than what they found, as coming from there. Others, from selfish or mercenary motives, reported what they hoped would bring emigration or raise the value of the lands they owned, so that little or no credit should be given to what was written previous to about the close of the last century. There is no truth in the statement that Sir Francis Drake found gold in California, which was then called New Albion, in 1579. No gold had been or has since been found anywhere in the vicinity of Drake's Bay, which is named after him, and which is the most northern point of his voyage. There is not the least probability that he had seen or even heard

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of the gold beaches which lie further to the north. There was, however, a very general belief that these countries contained both gold and precious stones as well. During the first half of the seventeenth century more than twenty expeditions went through parts of that country in search of them. The gold found by Captain Jediah S. Smith during his overland journey in 1826 was found in the Sierra Nevada Mountains, and not in California. One of the first authentic records of finding gold was near the Colorado River, in San Diego County, in 1775. But little work was, however, done at this place, as the ground was too poor to justify any serious exploitation of it.* The true history of the subject appears to be that towards the close of the year 1833,† or the beginning of 1834, J. P. L. Leese came to Los Angeles from Taos, New Mexico, bringing grain gold which had been washed out of the placers there. About the same time, a Spaniard named Palacios came to the same place from Guaymas, in the State of Sonora, bringing with him silver bullion in bars and bricks, and gold in grains from Sonora. Both lots together amounted to about $10,000 in value, and were for the most part taken out of the country, being carried by trading ships to Boston. A part of it was worked up by the jewellers in Los Angeles, but no tradition or belief that any part of California contained gold in workable quantities can be authenticated up to about 1841. In the early part of that year Don Andres Castillero, a Mexican gentleman of scientific education, went to Los Angeles and Santa Barbara, and saw what he believed to be evidences of the occurrence of gold near the ranch of Las Virgines, and brought with him specimens to Santa Barbara which he said indicated placer gold. In the month of June, 1841, Francisco Lopez, who had met Castillera in Santa Barbara, while riding over the ranch of San Francisquito, about thirty-five miles north of Los Angeles, found in a field of wild onions, small nuggets of gold which he took back with him when he returned to Santa Barbara. The rumour soon spread, and although very few of the Californians had any practical knowledge of gold-washing processes, there were a number of natives of Sonora, and other

* Report of the State Mineralogist of California, 1884, p. 217.

† Engineering and Mining Journal, vol. xxxii., p. 170.

parts of Mexico and California, especially in the central part of it, who had seen placers worked, and during the first two rainy seasons following, a few hundred adventurers, from San Diego to San Luis Obispo, flocked into that part of the country and were profitably engaged in winnowing and washing gold, which was sold at Los Angeles every year. During the first year Lopez is said to have taken out about $8000. The value of 18 oz. sent in November, 1842, to the United States Mint, at Philadelphia, was said to be about $340.

It is an undoubted fact that the Jesuit missionaries for many years previous to any reported discovery of it, had known that there was gold in the country, and placer deposits had even been worked in a small way at their station of San Fernando in Los Angeles County, but it was discouraged by the priests, who, believing that the passion for gold was demoralising to their people, taught them that the working for, and the possession of gold, was detrimental to their own interests, and contrary to the true principles of religion. The simple folk accepted the dictum, and but for the rumours which came to the East so repeatedly, and from such different sources that they could not be treated as myths, the gold might not have been discovered for many years.

A carpenter, named James Wilson Marshall, who had been attracted to California, and was working as a millwright putting up a sawmill on what is now known as Sutter's Mill, at Coloma, El Dorado County, on January 19th, 1848,* made the discovery of gold in the sluice of the mill which he was building, and subsequently in other places. Finding that the tail-race of the mill was not deep enough, he had adopted the plan of hoisting the gate and washing out the dirt with a full head of water, rather than dig the sluice deeper. On the morning of this day Marshall went out to superintend the men, and after closing the forebay gate, shutting off the water, walked down the tail-race to study the character of the sand and gravel which had been removed during the previous night, for he had conceived the idea that there were minerals in the hills, and he said so to Sutter, who had only laughed at him. As he was examining the débris, his

* Min. Resources, U.S., 1872, p. 111.

eye caught the yellow metal lodged in a crevice on a natural riffle in soft granite, some 6 in. under the water. After a close examination he suspected that this substance was gold. Being a shrewd man, who realised the importance of his discovery, he went on with his work as usual, showing the nugget to his men, however, in order that they might bring to him any such material as they might find. Making to himself some conjectures about the extent of the possible gold-fields, he watched carefully for any further developments, and in the course of a few days had collected a number of ounces of the metal. About four days after the discovery he had occasion to go to Sutter's Fort, now Sacramento; so, taking the grains with him, he examined the rest of the country, and again discovered the same metal in a ravine in the foot-hills, and at a place known as Mormon Island. Sleeping under an oak tree, he reached the fort, some ten miles distant, next morning, and soon put an end to all doubt as to the value of the discovery that he had made. It has been claimed that the gold was shown to Marshall by Peter L. Weimer, one of his men. Both men appear to have been together when the nugget was found, but Marshall, having a horse, went to tell Sutter of his discovery. Weimer gathered some of the small pieces afterwards, which he gave to his wife, who boiled them in strong lye, and as it stood this test without becoming discoloured it was concluded that it was gold. None of the three, however, knew with certainty that what had been found was gold, until it had been assayed in San Francisco. Marshall took his nugget to Sutter, who tested it with acid, and, presuming that it was gold, went back with Marshall to the mill, but kept the secret until the mill was nearly completed and stocked with logs. In the meantime one of the mill hands, named Bennet, had gone to San Francisco with some of the gold-dust, and showed it among others to Isaac Humphrey, who had washed in the gold diggings of Georgia. He pronounced it gold and started for the mill, where he arrived on the 7th of March. He at once made a rocker, with which he rocked the dirt of the mill race, and for several days washed out from 1 oz. to 2 oz. of gold a day. This result put an end to all work on the mill. The only use for lumber then, was to

build rockers, and nothing was thought of but getting the gold.

These discoveries were soon noised abroad, and caused the abandonment of the poor placers of the southern part of the country, and the influx of a host which depopulated whole villages and towns, the city of San Francisco amongst them, and caused the wildest excitement all over the East.

Although the discovery was so memorable, and the results of it were so important, it is hardly possible to-day to distinguish the spot where it was made. It has been proposed many times that a monument should be erected at Coloma, on the site of Sutter's Mill, but it has never yet been done, and it is not now likely that the spot can be distinguished with sufficient certainty to make it worth while to attempt to mark it, so that it is not probable that any monument will be erected.

Marshall died in Kelsey, California, on August 12th, 1885, in great poverty and distress. His cattle were killed and his claims jumped by men who hunted him, during the last years of his life, in the hope of discovering the secret of finding gold, which they believed him to possess. His life was made wretched by his disappointment in never finding the fortune he was seeking, and that he always supposed to be almost within his grasp, but which he never succeeded in obtaining. Some years before his death, the Legislature of California appropriated $200 a month as pension for him for two years. At the end of this time the appropriation was not renewed, and the man who had been the means of the finding of one billion six hundred million dollars worth of treasure in that State, died a wretched pauper.

The movement of emigration commenced about the middle of March. Before the end of the year over 10,000 persons had collected at the mines. This sudden influx caused the business of gold getting to be carried into other parts of the State. Pierson B. Reading, a ranch owner of Clear Creek, 200 miles from Coloma, having made a visit to the spot about the middle of March, returned home, and was extensively engaged in gold washing on his own ranch before the end of the month. John Bidwell, from the Upper Sacramento, having likewise made a similar visit, went to work on the bars of Feather River; and so the search for gold,

and also gold washing, extended to all parts of the State. Within three years after the discovery not less than 200,000 people, and within five years not less than half a million, had settled in the gold regions of the State of California to engage in gold mining. They had no laws applicable to their occupation or condition, and none were made for them, either by the State or General Government, in whom all the title of the lands occupied rested. As some organisation was necessary for their kind of life, custom and usage in each camp came, little by little, to have the force of law; but it was different in every camp, and resulted in the greatest confusion. It was not until after eighteen years that the United States commenced to legislate in regard to mining, and this neglect on the part of the General Government has been the cause of untold expense in litigation.

What these men really earned can be easily calculated. From 1850 to 1854 there were at least 100,000 men engaged in mining gold in California. The largest amount ever produced in a year in the State was $65,000,000.* If the men generally worked two hundred days in the year the average daily earnings were less than $4. It cost at least $3 a day to live, so that the net earnings were in these days about what they are now. We know the enormous fortunes made by the few, so that the multitude must have earned very little. These few fortunes were heralded far and wide, but no one will ever know the misery, suffering, and despair of the immense number of most unfortunate ones who died disappointed, embittered, or insane.†

Previous‡ to the military occupation of California by the United States Government there was but little money in circulation, and but little was needed, as trade was conducted almost exclusively by barter. Ships came occasionally from the United States. They were always known as "Boston ships," from whatever part of the country they hailed. They brought assorted cargoes which were bartered for hides and tallow. The Hudson's Bay Company carried on their trade in the same way. The trade with Peru, which consisted mainly of sugar and French and Chinese silks, was conducted for the most part by barter, but money was sometimes

* Vol. i., p. 35. † Vol. i., p. 3.

‡ San Luis Obispo Tribune, May 20th, 1887.

required. The occasional direct importations from China had always to be paid for in coin. The duties on the merchandise from these two last places alone depleted the country of its coin. The only way in which coin was brought into the country was by custom-house dues, which were often not sufficient to pay the expenses of the Government. It is estimated, however, that more than half the duties which should have been paid were, owing to official corruption, never collected. The average amount of payment of duties from 1836 to 1846 did not exceed $100,000, and this money was distributed as fast as received for Governmental purposes, and a large part of it went to China and Peru.

When the news of the gold discoveries reached Mexico, Chili, Peru, the Sandwich Islands, and China, valuable cargoes were at once sent out from there. The merchants of San Francisco found themselves in a very anomalous position. They had merchandise afloat in the bay, which they could have sold for a very large profit if they had had it ashore, but they could not land it, as it was impossible to procure a few thousand dollars in coin with which to pay the duties. It was found on investigation that it was more than probable that the total amount of coin in circulation was not sufficient to pay the duties on the merchandise then afloat in the harbour. The merchants represented to the collector of the port that, under the circumstances, for which they were in no way responsible, it was the duty of the Government to make a liberal construction of the law requiring the duties to be paid in coin, and as there was not in the whole country coin enough to satisfy the Government demand, that gold-dust, certified to be worth $16 an ounce, should be received in lieu of coin for duties. After much deliberation, both for fear that he might be held responsible in Washington for receiving gold-dust as coin, and to cover himself against any possible pecuniary loss in the value of the gold, the collector decided to receive gold-dust worth $16 an ounce as equivalent to $8 in coin. This settled the question so far as its effects on the ships in the harbour were concerned, and a nominal purchasing power was given by the authorities to gold-dust. Immediately after this decision, however, the small tradesmen held meetings and declared that they would only receive the dust at the value put on it by the Military

Government, and that gold-dust to be received by them at $8 must have the value of $16. The merchants, however, seeing the folly of this procedure, determined to and did receive the dust at its full value of $16, and thus broke up the combination, but so phenomenal was the quantity of gold-dust sent in from the mines that many of them believed, with some reason at the time, that the value of gold would be so lowered that silver would have to become the standard of value in the country. This state of things continued until the arrival of the Panama steamers in 1849.

Up to this time it had been almost impossible to establish relations with any other part of the globe. People came from everywhere, but no one went back. There was neither the means nor the desire to return. Ship after ship came through the Golden Gate, but no ships, once in the harbour, went out of it. Each vessel as it arrived would be stripped, the sails unbent and stowed away, and the lighter spars sent down on the deck. It would then be moored to swing with the tide. From the captain to the cabin-boy, every one knew that the voyage had come to an end. The crews in every instance, and even the officers in many cases, deserted, taking the ship's boats and provisions, to sail up the Sacramento to the diggings. Valuable cargoes were thus often left for weeks guarded by a single storekeeper, put on board by the consignees. This fleet became so large that at one time it seemed as if the whole bay would be covered with ships, so that there would no longer be room enough even for mooring other vessels should they arrive. Under these circumstances, as no one went out of the country, the gold accumulated in vast quantities. It simply poured into San Francisco and found no outlet. In the last months of 1848 some remittances had been made through English men-of-war to Panama, Callao, and Valparaiso, but this did not amount to anything in its influence on the whole country. Everybody had more or less of the dust. Thievery and beggary were unknown. There was so much gold-dust that everybody had it and nobody wanted it. Merchants would leave valuable merchandise, even in portable packages, at the landings for weeks, and nothing was ever missed. Skilled labour was worth $10 to $12 per day; unskilled labour,

$6 to $8 per day, and $1.50 to $2 per hour. · Clerks received $250 per month and their board, and cooks were paid $200 a month. In the winter of 1848 beef sold for 50 cents per pound, and cabbages from $3 to $4 each. Wholesale merchants demanded from 100 to 250 per cent. advance on all invoice prices, and retail dealers often more than this, so that general merchandise cost the consumer from 225 to 500 per cent. advance on first cost. The price of the commonest clothing was from $60 to $150 a suit; boots, $12 to $20 a pair; flour, $26 a barrel; ham and bacon, $1 a pound; lard, 75 cents; and butter, in kegs, $1 a pound. Notwithstanding these enormous prices the market was always well supplied, and all articles found a ready sale. As gold became more abundant the prices advanced. This fell hardest upon the officers of the army, whose pay was not increased, and who were obliged to keep up a certain appearance, notwithstanding the unfavourable circumstances in which they were placed. There were no servants of any kind to be had. A collector of the port at the time took his turn as a cook, and his successor had to learn the trade of shoemaker to get his own boots mended. The custom house and the gambling houses absorbed any money as fast as it came into the country, and eventually the price of gold fell to $6 the ounce, and then fluctuated between $6 and $11. Merchants borrowed of the gambling houses at the rate of 10 per cent. a month, payable in gold-dust at $16 an ounce. In August, 1849, the collector of the Port of San Francisco advertised a sale of 11,000 oz. of gold-dust, which had been deposited for duties. It was sold at public auction for $11 an ounce. It had been received for duties at $8. The difference between the two, $3, was retained by the collector to reimburse himself for the expenses of the sale and for the safe keeping of the dust between the time of its deposit and sale.

The arrival of the Panama steamers at once put an end to this anomalous state of things. They brought coin from Mexico, Peru, and Chili, to purchase gold-dust, and it consequently advanced steadily in value with the demand, and commenced to pour out of the country as it had poured into San Francisco, until the amount exported monthly reached $5,000,000. In the last part of 1849 there was even a slight stringency in money

after all this plethora. A fever of land speculation then broke out, and the demand for money was so great that 10 per cent. a month, compounded monthly, was the regular bank rate on prime paper or real estate as security, paper being preferred. The interest finally went up to as high as 14 per cent. a month. In 1850 the very large importations of coin from Mexico and Peru brought the dust up to $12 an ounce. Interest went down to 5 per cent. a month. When, in 1851, the United States Assay Office, then just established, issued $50 octagonal pieces, 900 fine, which were receivable for custom dues, interest fell to 2 per cent. a month, and then steadily declined, so that when the branch mint was established in 1854, gold-dust was sold at its real value and money fell to 1 per cent. a month, which had been the regular legal rate previous to the discovery of gold. Gold-dust then rose to $16. In 1851 the merchants of San Francisco made an ineffectual attempt to have the duties previously paid under the decision of the collector who had received the gold-dust at $8 refunded to them. This discussion created some interest at the time, but it failed to get before Congress, and the matter, in the press of other things, as the duties to be refunded amounted to only a million of dollars, was lost sight of and forgotten. The great influx of population in the latter part of 1849 produced great changes in business matters. One of the principal of these was the release of the fleet which had been confined so long in the harbour of San Francisco. The Golden Gate was thrown open for their exit, and signal flags for sailing were once more seen as the ships left, one after another. But they were obliged to go away in stone ballast, as there was nothing but gold-dust produced in the country, and this was sent to pay for the return cargoes.

The people who flocked to California under the conditions just described were not ordinary immigrants. They were mostly men in the prime of life, active, energetic, enterprising and intelligent, and they produced an entire change in the condition of the population of California. Previous to this time only uneasy and restless men, mostly patriots who left their country for their country's good, had come across the plains to California. They found the government exactly what it had been under the old Mexican system,

and at first were too busy in seeking their fortunes to make any change in it, but as soon as this was demanded the change was made, little by little, first in the mining camps, and then gradually extended over the entire State, but the feverish excitement, caused by the desire of every one to acquire great wealth in a short time, prevented a proper organisation under a State Constitution until 1850, when the first legislature was convened at San José.

Placer Deposits.

When gold is found in sand or gravel, the deposits are called placers. These are formed by the accumulations of the débris washed down by ancient rivers, and the mechanical and chemical deposition of gold in them often forming deposits of considerable depth and variable richness, which have themselves sometimes been again torn up and redistributed by more modern streams, forming deposits of comparatively recent origin. We have thus the ancient and the modern placers. They have, also, sometimes been either wholly formed, or, in some cases, only partially, by the precipitation of the gold dissolved in the waters of filtration, by organic matter. According to the depth at which the gold is found, they are called shallow or deep placers. The shallow placers are those deposits which extend to the depth of about 10 ft. or less from the surface. It was in these deposits that the gold was originally found in California, and as they are usually very rich, they are most eagerly looked for in the early settlement of a country. They existed, scattered over a great extent of territory, and were so rich and so largely worked that in the first five years after their discovery they produced, notwithstanding the wasteful and primitive methods which were used to work them, $180,000,000. They are not always of the same richness and value, nor are they always capable of being worked. Abundance of water is the chief requisite to their successful treatment, and even then they are often worked on a wrong theory, and only while the pay dirt continues very rich, and by the rudest and most wasteful methods, much more being lost than was ever extracted. They were often abandoned as worked out before their real richness was discovered, some of

them having been thus worked over two or three times in succession. Even now the Chinese make a good living by working over these old tails.

The methods used by the pioneers of California are called mining, because they eventually led to most systematic and ingenious processes, but they would not be tolerated now. Their methods would be justly termed "robbing," yet, but for the efforts of these men, who would never have accomplished anything if nature had not previously done most of the work herself, the great precious metal deposits of California might still lie buried where nature placed them. While they left in the ground as tails most of the wealth which they worked at, but for their rude methods we might never have been forced to work out the problem of treating the vast treasures of poor sands and ores which they left behind them.

The days of successful haphazard mining have for ever gone. To work out the great problem of developing the vast resources of the country which are entombed in its poor and rebellious ores, capital, mechanical skill, and science must work together, and these must be in greater proportion as the ore is more refractory. These are the days of the highest development of engineering on a large scale. No man and no company can hope to be successful without grappling with the most intricate problems of engineering, metallurgy, and mechanics.

Up to the time of the invention of hydraulic mining, the shallow placers furnished nearly the whole of the gold produced in California. They were abandoned because, with the primitive means in use at that day, they were no longer profitable, but they are by no means exhausted. It is not improbable that it will, in the near future, be found profitable to work them over again on a large scale, both because the gold was only partially extracted as well as because it will, probably in a few decades, be again sufficiently concentrated in them from precipitation of the minute quantities of gold contained in the waters of infiltration.

It is estimated* that since 1851 the total amount of gold produced in the world has been $4,500,000,000. Of this sum California has produced about $1,125,000,000, and of this amount

* Prod. Gold and Silver in the U.S., 1881, p. 616.

$1,000,000,000 have been produced by the working of the auriferous gravels, the largest part of which came from Nevada County alone. The amount extracted was but a small part of that which was originally in the sand. Only a very small portion of the gold-producing country of California has ever been worked, and the wealth of gold as yet extracted from the soil is but a very inconsiderable portion of that remaining in it.

Not unfrequently small particles of the precious metal were found in the roots of the grass or of small shrubs torn up by accident or design. Sometimes the shining particles of gold were seen in the streams whose course lay through the ancient deposits or received the drainage waters from them. It was not until long after these modes of occurrence of gold had become known that it was discovered that the precious metal might be found deep down in the earth on the bedrock of ancient rivers which had been covered with detritus to the depth of hundreds of feet. These modes of occurrence give rise to many methods of exploitation of gold, the principal of which are known as placer and vein mining. The placers are divided into shallow and deep placers, and the method of mining varies, according to the manner in which the gold occurs in each, as follows:

In Shallow Placers.

Gravel mining.
Sluice mining.
Beach mining.
River or bar mining.

Deep Placers.

Hydraulic mining { Surface mining. / Drift mining.

Vein Mining.

Dry crushing } Amalgamation
Wet crushing }
Concentration and chlorination.

Gravel Mining.

While the shallow placers are at, or very near the surface, the deep placers, as the name implies, have sometimes their richest portions near the bedrock, which is often 200 ft. to 300 ft. below the surface, at which point the ground is generally quite poor. The formation of both these classes of deposits

is owing to the same general causes. In many cases the shallow placers are formed by the destruction of pre-existing veins, and the accumulation of the gold from the detritus by the action of the waters. In others they have undoubtedly been formed by the infiltration of small quantities of gold in solution in water which has percolated through them from the surface, and has been thrown down in the sand by organic matter, either dissolved in the water or scattered as solid particles through the gravel.

The methods by which shallow placers are worked differ from each other more by reason of the differences in the occurrence of the gold than because of any difference in the principle of the methods employed to extract it. The shallow placers are frequently divided into wet and dry diggings, wet being those which are found in the bars and beds of streams, dry being those which are found in the gullies and the bottoms of the streams, which are large in some seasons of the year and dry up at others, so that water can only be had during certain portions of the year. These last claims, where the water is transient, are sometimes called "gulch claims." They are also sometimes called "dry diggings," and are not unfrequently very rich in gold. All these were worked in the early days, but as they have ceased to be very profitable, they are now left to the attention of Chinamen. The early miner required but few utensils. His "outfit" was what he could easily carry in his hands and on his back, and was necessarily reduced to a minimum. His principal capital was a strong constitution and a stout heart. What he could not carry was of no use to him, however valuable it might have been or might be under other circumstances. Hope, which found no discouragement even when "bedrock" was reached, a light heart, his pick, shovel, and pan, and water to be found when the "pay" was rich enough, tins to rudely cook his food, and a blanket to cover him, were his investments and what he hoped to find, his rolling capital. Later on, when the diggings became further removed from centres, his equipment was not so simple, as he was obliged to provide himself with pack animals to carry his tent, tools, and provisions, so that his absence could be prolonged until he found the "pay." He never considered the

diggings as his home. There was always some place to which he hoped to go when he "struck it rich," and no discouragement or failure could take this ideal home of his away from him.

In the very early days the pan was almost the only instrument used. The pick and the shovel were the allies of the pan, but as they were only indirectly the means of getting the gold they were not considered so important. These two indispensable tools never found a place in his vocabulary, nor seemed to have the hold in his thoughts that the pan did, which still holds its own as a prospecting tool, although it has lost its importance as a working instrument, as it is by far too expensive in time.*

FIG. 1. THE PAN.

At first it was any household pan, Fig. 1, that could be made available. Now it is made of Russia sheet iron, but as its shape is such as is convenient for household purposes, the same pan is used for one and the other. It generally holds about 400 cubic inches of loose dirt. Upon its skilful manipulation the success of the miners depended, and they soon learned to use it so that in the rich deposits, or even in the poorer ones, they handled it with almost artistic skill. The work performed by it was small in amount, but so certain in its results that

* Vol. i., p. 16.

its revelations were regarded as infallible, and districts were overrun or abandoned according to the results which it gave. In studying the method of its use the work may be divided into periods, which will generally be found to correspond to five different phases or sets of motions. First, the pan is filled with the pay dirt to about two-thirds of its capacity; it is then, according as water is abundant or scarce, placed in the bottom of a shallow stream, or set in a hole filled with water. It is usually placed on the bottom so that the operator can use both his hands. Occasionally it is held in one hand while the work is done with the other. The whole contents are thoroughly stirred with one or both hands. Any lumps that may be found are broken and mashed between the fingers, so as to mix them all thoroughly with the water. When the whole contents of the pan are in the condition of a thin homogeneous mud, the first part of the operation is finished. Second, the pan is now taken in both hands, a little behind its diameter, raised nearly to the level of the surface of the water, but a little below it, inclined away from the operator, so that a very slight movement disengages all the light matter and causes it to pass over the edge of the pan. This is continued until the water which passes off is perfectly clear. There will remain in the pan, after all the lumps of clay have been broken up, only pebbles, gravel, heavy sand, usually black, and gold-dust. The third stage commences with raising the pan above the water, picking out all the large pieces, and examining them before they are thrown away to see that they do not contain gold. The pan contains but little water, so that it is quite easy to get rid of the large pieces of sterile matter, either picking them out or pushing them over the edge. There will be now in the pan little else than small-sized pebbles, the black sand, and the gold. The fourth stage is to incline the pan a little under the water, causing the next smaller-sized pieces to pass in such a thin stream over the edge of the pan that every particle is visible, so as to be sure that no gold goes over. The fifth stage consists in lifting the pan out of the water with a little water in it, and, by a dexterous movement of the wrist, causing the sand to pass with the water across the bottom of the pan, leaving "the colour" on the opposite side. The pan is then

inclined to the side where the sand is, and this is washed out with small quantities of water taken up in the hollow of the hand and poured on it, causing it to pass out, leaving the gold behind. The black sand, however, still remains with the gold. There are three ways of getting rid of it. The first is with a magnet when the sand is magnetic; the second, by dexterously blowing away the sand after it has become dry, and the third by the use of mercury. The first two are the methods most generally used.

The ordinary miner's pan has a flat bottom; the Mexican and Brazilian modification of it is called a *batea*. It is generally a round piece of wood turned in a lathe, and is about 20 in. in diameter, conical, and not over 3 in. deep in the centre. It is sometimes made of beaten or spun metal. It is in every way superior to the pan, and would have been universally employed if it could have been obtained. It is not much more difficult to manipulate than the pan. It is much used now, as it has been shown that the results of the assay are much more quickly obtained on a wooden than on an iron surface, and it requires only a little more skilful manipulation. The early miner preferred the iron because he had it. He did not have the *batea*, nor would he have known how to use it if he had. On the principle of this implement some successful prospecting and working tools have been made.

Great use was made in the early days of the *horn spoon*, which was cut out of a large ox-horn so as to make a sort of spoon about 1 ft. long by 2 in. to 3 in. wide. Horns of dark colour, or those which were black at one end at least, were much sought after for this purpose, as the gold is so much more readily seen on a dark than on a light surface. It is much used for testing samples and is much better for this purpose than a metallic surface, because it does not readily become coated with grease, and because air does not become attached to it as it does to a metallic surface, and thus prevent its becoming wet.

All of these were rather prospecting tools than mining implements. They were, and still are, used in all new fields, both as prospecting and working tools while the pay dirt is rich, but when it becomes poor, so that the quantity to be treated in a given time must be great, they are no longer capable of treating the large

amount necessary to give a fair return for the labour, so that if water was not abundant, the field was abandoned, or if it was, "machines" were substituted for the pan. The first of these used was the "cradle," or "rocker," Fig. 2, as it is often called. This was a box on rockers, having a grating on the top 20 in. square, called the *riddle*, on which the gravel was placed. Water was poured on this, which carried the pay dirt through. It fell on an inclined blanket-apron, which carried it to the back of the lower division. On this lower part are two riffle-bars, about 1 in. high. The cradle is slightly inclined to the front, so that all the dirt and

FIG. 2. THE CRADLE OR ROCKER.

water put in at the top ran down the apron to the back, over the bottom, and out at the front. The weight of the water used in the cradle is from three to four times the weight of the dirt to be treated, so that the work must be done near the water. The very fine gold is caught on the blanket apron, the coarser particles behind the riffles; the larger stones are easily picked out from the riddle, and the small ones are left to help break up the tough dirt. The motion of the cradle must be constant to dissolve the dirt and keep it loose; otherwise it would pack hard and the gold

would run over it and be lost. The amount of water used must be large, or the excess of dirt would not be carried off. When the blanket is to be cleaned up the riddle is removed and the blanket taken out and washed in a pail. All that is caught has to be panned. If the rocker is to be worked by one man, the water is collected in a hole made at his right hand. With the left he moves the rocker, and with the right he pours water over the dirt. If the "pay" is near the water it is brought in pails or wheelbarrows. If this cannot be done, it is "packed" on mules or brought in wagons. If worked by two men, one brings the water and the other rocks it. If the water can be brought by a

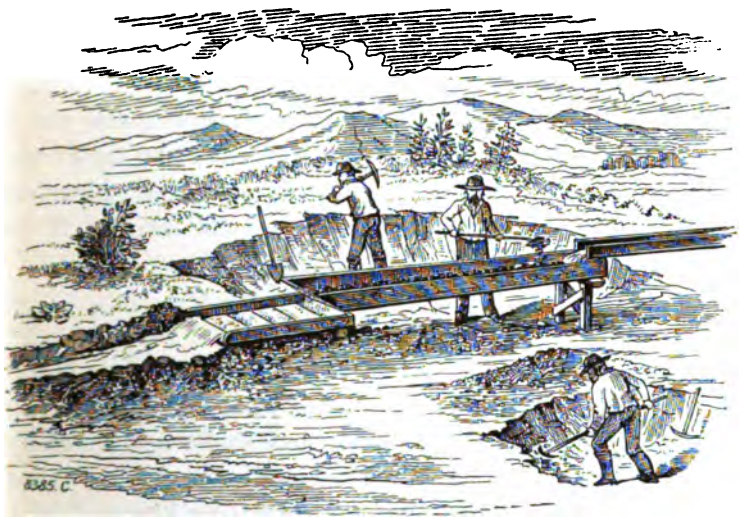


FIG. 3. THE "TOM."

sluice two men can work rapidly. One man can treat one and a half cubic yards in a day; two men can do nearly three times as much. When the gold is fine the cradle, though very cheap, is not economical, as most of this gold is lost. When it is coarse and the supply of water uncertain, it is a very useful machine; but its output is very small, and the amount of labour required to be done with it is very large. At the best it can only treat half the amount that can be washed in a "tom" with the same amount of labour. Its use has been abandoned in this country, except by the Chinese.

The rocker was replaced by the "tom," Fig 3, which was really

only two launders or sluices, and required a supply of water from another sluice. The apparatus consists of an upper sluice-box about 14 ft. long, 20 in. wide at the upper end and 30 in. at the lower, with an incline of a little less than 1 in. to the foot. The lower end of the launder is cut off at an angle of 45 deg. This is closed by a grating of punched sheet iron, which allows everything but large stones to pass. This ends over a riffle-box, about 3 ft. wide and 11 ft. to 12 ft. long, at the same incline, with five or more riffles, into which mercury is put. One man throws in the dirt, another stirs it, breaks up the lumps of clay and dirt, and removes the large stones. The gold is caught in the riffles, which are sometimes supplemented with blankets. It is worked by from two to four men. Two men will do five times as much with the "tom" as with the rocker. It is never worth while to use the "tom" except when the gold is in coarse grains and the quantity of pay dirt small. It is now only used by the Chinese, who use it to work over the "exhausted" tails of the abandoned workings of the early miners, and are quite satisfied if they only make 75 cents, or even less, a day.

Sluice Mining.

Almost all the shallow placer deposits are now worked by sluices. These sluices are tools of great value and simplicity, and their invention and use has probably done as much to increase the wealth of California as any other one implement. They were for many years the most important instruments used in working the placers. The sluice was invented in California, although it had been used elsewhere, under, however, very different conditions. It has been brought to its greatest simplicity, as well as its greatest perfection, in this country. It is made of rough unplaned boards, and is of almost any length over 50 ft., and is not less than 8 in. wide and 9 in. deep. The width is usually 16 in. to 18 in.; it may be thousands of feet in length. The boards are 1½ in. thick and 12 ft. long, just exactly as they come from the mill. The sluice is made in sections. Each section is called a "box," and as the boards are from 12 ft. to 14 ft. long, the sluice is said to be made up of so many "boxes," which indicates the length perfectly. The sides of these "boxes" in very large sluices are

sometimes from 18 in. to 2 ft. high. The bottom boards are sawed for this special purpose, and are 4 in. wider at the lower than at the upper end. The narrow end is intended to fit into the wide one of the upper end of the next one below. The sides are nailed upon the bottom. No attempt is made to make them tight, as the joints become filled with sand or swell tight after a little use. The sluices thus made rest upon trestles with a descent varying from 8 in. to 18 in. in a box. This descent is called its "grade," and if the fall from one end of the box to the other be 8 in., it is said to have an 8-in. "grade." It is generally not desirable to have a grade of less than 8 in. or more than 20 in. The boxes are easily put up on trestles and readily taken down, and are transported from one claim to another after the placer has been worked out. The amount of water to be used in a sluice will depend on a great variety of circumstances, especially the quality and quantity of pay dirt to be treated in a given time. To work successfully there should be plenty of water In general the depth of water on the bottom of the sluice should not be less than 2 in.

The water for the sluices is obtained in various ways. If the stream which brings it is very small, too small to furnish the water required for a day's working, it may be dammed and the water used as long as it lasts. If the stream is a little larger it may be turned into the head of the sluice, using a gate to regulate or cut off the supply; if it is too large a race may be built to supply the requisite quantity. Occasionally the claim may be situated near the banks of a river where a "flutter wheel," Fig. 4, may be put up in the bed of the river and the water for the sluice brought up by it. The construction of these wheels is very simple. They can easily be built when there is wood enough in the country. They do the work very well, though they are very wasteful in power, an item which is of no consequence, since all the power would be wasted if the little used for raising the water was not utilised. In the early days of placer mining such devices were very largely used. They have almost entirely disappeared at present, owing to the ease with which water can be obtained from the ditches and the abundance of it, as well as the decay of this kind of placer mining, which has died a natural death, owing to the concentration of capital and the ownership of large areas

of ground, which must be worked on an enormous scale in order to bring any profit. The grade depends upon the character of the "pay," as the dirt containing the gold is called, the length of the sluice, the character of the ground, and the quantity of water available. The steeper the grade the more rapidly the dirt will be worked, but the greater likelihood there will be of loss in gold. Ordinary dirt will be entirely broken up within from 200 ft. to 250 ft. of the place where it is charged, and from this point, the sluice has but one function, namely, to catch the gold. The place being determined at which a given quality of dirt will be completely broken up, the grade beyond that point

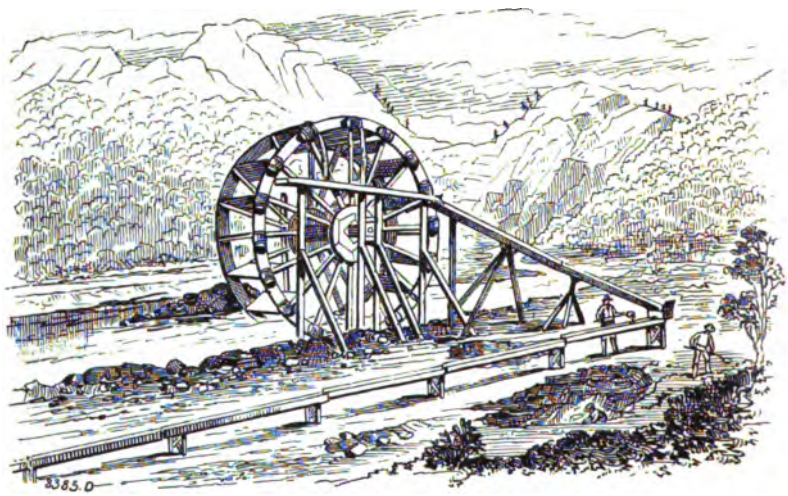


FIG. 4. THE FLUTTER WHEEL.

is usually lowered, but must always be sufficient to carry off the excess of material. It is rare that a sluice is made long enough to catch more than 50 per cent. of the gold contained in the sand. Tough, clayey material requires a steeper grade than any other. To break up this tough clay the flow must be rapid, but in such a case the sluice must be long. The grade must always be less in a short sluice than in a long one. This is owing to the fact that there is always danger that a rapid current over a short distance will carry off the gold, whereas in a longer distance it would probably be caught. When the amount to be treated is large the grade should be high, the water abundant, as also when

the ground contains a considerable quantity of large boulders. It is unusual for the sluice to have the same grade throughout. It is generally made steep at first, so as to break up any tough material, and less steep afterward. Usually, however, the miner makes the grade conform to the lay of the ground, whatever that may be, and regulates the height of the trestle by the ease with which he can throw the dirt into the sluice. There is always danger, when the dirt contains any plastic clay, even when the grade is very high, that this will roll over and over on the bottom, and take up the gold, which has already been deposited, and carry it out into the tails at the end of the sluice. Such balls may sometimes roll a thousand feet without breaking. For this reason special care is taken to make the sluice for this kind of work very steep at first to break up the clay, and lower afterwards to catch the gold, or, better still, no attempt is made to work the clay in the sluice; but it is first puddled either by hand or machine, and not introduced into the sluice until it is in the condition of liquid mud. For this reason when clay balls are found in the sluice they should at once be removed, broken up, and puddled. When there is but little clay hard work is all that is necessary to break up the clay in the sluice, but when there is much of it, it is cheaper to puddle the whole than to run the risk of losing large quantities of gold. When there is a great deal of gravel or large stone to be removed, under currents are used. For this purpose the end box is left open, and the bottom, for the width of the sluice, is covered with a grating sufficiently coarse to allow all the fine stuff to pass through. Under this a short sluice at right angles to the direction of the main sluice is placed, which ends over another sluice parallel to the main one but at a lower grade. The material coming from above, which is fine enough, passes the grating and goes on in the other sluice. The stones and coarse stuff fall out at the open end by the force of their own velocity.

The gold is caught in the lower end of the sluice by various devices, which consist of the insertion of some kind of false bottom. Even though the boards are rough, with their splinters turned up stream, they would soon be worn smooth by the material passing over them, and the gold be swept away in the current. In order to prevent both the mercury and the material from run-

ning out of the box too quickly, narrow strips of board or of iron are fixed on the bottom. These strips are called "riffles." They may be either transverse or longitudinal, or both. The longitudinal riffles are from 2 in. to 4 in. thick, 3 in. to 7 in. wide, and 6 ft. long. From two to four are placed on the bottom of the box. Each box will, therefore, contain two sets of such riffle-bars, and whatever the sluice contains, whether two or more, is called a set. They are wedged to their places, and held together by a certain number of transverse riffles, so that they may be easily removed for a clean-up. It would not do to put them in permanently with nails, as they must be often taken out. Immediately after a clean-up the riffles are adjusted in the boxes. There will thus be in each box, with longitudinal riffles, sections which will be 6 ft. long, and the length and depth between the riffle-bars. When the sand is fine, transverse riffles, which are strips of wood 2 in. by 4 in., wedged in at various distances across the sluice, are often used alone, but when there is a small amount of pasty material to be broken up, zig-zag riffles at an angle of 45 deg. are sometimes used. These riffles reach nearly across the sluice and are nailed to the bottom. The water, with the fine metal, passes over them, while the coarse gold and the clay go from side to side with considerable velocity. Such riffles do not catch the gold, but only help to break up the clay. They are placed in the upper part of the sluice, which must always be terminated by the ordinary riffles. Occasionally the sluice has a wooden pavement, made of blocks cut across the grain of the wood and set several inches apart, but so placed that they can be easily removed. They are not so permanent as a rock pavement, and are not much used except in hydraulic mining. Sometimes, where the works are to last for a considerable time, the sides of the sluice are protected with a "lining" of boards two-thirds the depth of the sluice, so that they will not be too rapidly worn.

The coarse gold is easily caught at the head of the sluice, but much of the fine gold would be lost if nothing further was done to catch it. For this purpose mercury is put into the head of the sluice, being poured through a strainer to divide it into small streams or in some other way to break it into small globules. It is usually not added until from one to two hours after the com-

mencement of the working. It is caught in the riffles and amalgamates the fine gold. Sometimes the pores of the riffle-bars are impregnated with mercury by introducing a gas-pipe at one end, and filling it with mercury to a considerable height. This is very troublesome and scarcely pays for the work it occasions, as the riffles must be cleaned by scraping from the wood the amalgam formed on them. It gives as much trouble as copper plates, and is much less effective. Copper plates are not often used in placer working. They are extensively used in battery sluicing. When they are used they are amalgamated by cleaning them first with dilute nitric acid, which removes the dirt, and then rubbing them with mercuric nitrate, or with mercury alone, which is rubbed on with a piece of cloth. When the former is used the nitric acid brightens the plates, and the mercury at once amalgamates with the copper; the green slime of basic copper nitrate thus formed must be carefully rubbed off. It is much quicker to clean the copper with potassium cyanide, but this requires more care, as if too strong a solution is used, or the plates are left in it for too long a time, they become black and will not amalgamate. When plates are used the current must be slow and the water shallow. A swift current carries off the fine gold and deep water does not give the gold time to settle. When a speck of gold settles on any part of the plate, others will collect around it. The more gold there is on the plate the better it catches the gold, so that when the miners can afford it they amalgamate the plates with gold; when they cannot afford that, with silver, and where that cannot be done with mercury alone. The amalgam is left to accumulate until it is from $\frac{1}{4}$ in. to $\frac{3}{8}$ in. thick, but not thicker, as there is danger of its being stolen. It is then in a hard mass, which must be separated by force. To do this the plate is heated until it is a little hotter than the hand can bear, and the amalgam separated by scraping or with a chisel. A copper plate $\frac{1}{10}$ in. thick, after it is once amalgamated, can be made to last from five to ten years if treated with care. An amalgamated copper plate is better than a bed of mercury of the same size, but it must be kept clean by rubbing. This is usually done with an india-rubber tool, not unlike the one so widely used for cleaning windows. The time

between two clean-ups is called a "run," which usually lasts from six to ten days. The sluice is generally run only in the day, but is sometimes worked day and night. Men must be kept on the watch to see that the material coming down does not choke the sluice so as to make the water pour over the sides. Whenever a dam commences to form they break it up with an iron bar and throw out the stones with a fork.

Sometimes the sluices are made double by running a partition through the centre of a large one. This is done either when two companies are working on the same ground, in order to avoid the extra cost of two separate sluices, or when, because of the variable supply of water, it is desirable to run double the amount, while the water lasts.

Sometimes, in order to prevent unlawful clean-ups at night, the sluices are paved with stone. These paved sluices, however, are not usual except in hydraulic mining. An active man can pave in a day 800 square feet of such a sluice of cobble, with stones 6 in. to 8 in. in size. They are laid in loose, but after the sand has run over them from half to three-quarters of an hour they will be packed quite tight. They are difficult to lay and to clean up, but are much more permanent than the wooden pavements. A patrol with a shot-gun loaded with bird shot, and a plentiful supply of salt and pepper in the charge, is a much better protection against theft. No one who has ever heard of the pain caused by the reception of such a charge, in any part of the body, would ever attempt to clean up for his neighbour, if he supposed that he might possibly suffer this punishment.

The gold, quicksilver, and amalgam are caught in the spaces between the riffles. All the mercury is introduced into the top of the box and passes down with the rest of the material. The water should be at least 2 in. deep over the bottom. Boulders and stones must not be allowed to remain in the sluices, as they not only grind up the mercury and amalgam into flour, but are in danger of becoming wedged and of obstructing the free flow of the water. They are taken out along the sluice whenever they are observed, by the man whose duty it is to watch for them.

Sometimes a sluice is made without any boards by making a channel in the earth into which a stream is run. This channel

is called a "ground sluice," Fig. 5; it is either in the pay dirt itself or in barren ground, the uneven bottom of which makes catch places for the gold. No other means of catching the gold is used except that afforded by the uneven bed of the sluice. It was used almost exclusively in the rainy season when water was plenty and the ground soft, so that the banks of the pay dirt could be easily forced into the stream. As the water was very uncertain they usually worked day and night with pick, shovel, and crowbar, and abandoned the place when the water gave out. When a clean-up is to be made in "the ground

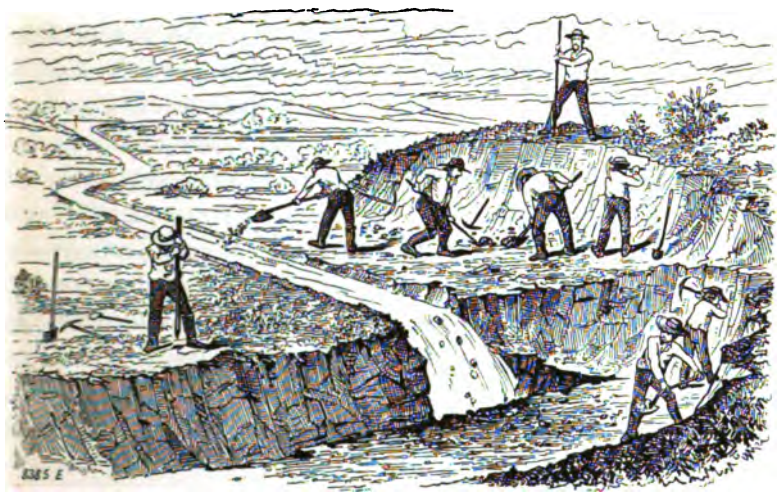


FIG. 5. THE GROUND SLUICE.

sluice" the water is turned off, and what is collected on the bottom panned. Only the coarse gold is saved; the fine is entirely lost. As it uses more water for the same amount of ground treated, it is only used in temporary workings.

When the rich dirt is below the level of drainage it is washed in a pit. There are two ways of doing this, depending for their application on the lay of the ground. One is to pump the water used in washing out of the pit with a Chinese pump, and the other to make a covered tail-race through which the water can flow to a lower level, and cover it as the work progresses with the washed ground. This generally necessitates the use of a short sluice, or even only of a "tom."

An active man can throw from three to seven cubic yards of dirt a day into the sluice, provided he does not have to lift it more than 4½ ft. to 5 ft.; at a less height he can do a little more. The quantity depends also somewhat on the character of the dirt which has to be moved. Sand without clay or boulders is most easily washed. Six cubic yards daily per man is fair average work. In a ground sluice where there is no lifting, but only shoving with the shovel to be done, the average work per man will be about 25 cubic yards per day. When a wooden sluice is worn out it is burned and the ashes treated for gold. If it has been used for some time the amount of gold recovered in this way will sometimes be sufficient to purchase a new sluice.

To make a clean-up the dirt is no longer put in at the head of the sluice. When the water runs entirely clear, five or six sets of riffles are taken up at the head of the sluice, their contents washed down to the riffles below. The sand is washed over, while the gold and amalgam lodge at the riffles and are removed. The next sets are then removed, and so on; sometimes twenty or thirty sets of riffles are taken out at a time. The work of a clean-up will generally take half a day to a day. It should not be made frequently, as it consumes so much time. It is generally done on Sunday. The length of the run will depend on the richness and character of the dirt. The amalgam collected is strained through a cloth and retorted.

When the dirt is rich and water cannot be had, the process of "winnowing" is used. For this purpose all the stones are picked out and the dirt is made as fine as possible. It is then tossed by two men on a hide or a blanket, with the object of having the fine particles carried off by the wind. After this has concentrated the pay dirt somewhat, it is again tossed on a batea. Where the pay is below the surface, small shafts are sunk and galleries run short distances in all directions from them. This is called "coyoting," from its resemblance to the burrows of the coyote. None but the richest pay can bear this kind of work. The material brought to the surface is "winnowed." Winnowing machines are now being made for use. The loss is enormous, but where they are used, what is obtained is all gain, for without this method nothing would be had.

In very many of the deposits, after the loose sand has been worked, there remains below it a hard mass which was recognised as containing gold, which could not be extracted. At first this was sent to the stamp mill, but as all of it did not contain gold, and all the coarse boulders which were sterile had to be crushed so as to pass through the screens, as well as the fine material which contained the precious metal, this method was abandoned. Most of this hard material, which is called "cement," disintegrates when it is exposed to the air for a certain length of time, or is broken up by the frost. The cementing material is usually iron oxide, or clay. It was at one time proposed to work it by the hydraulic method, interposing in the course of the sluices a large number of "drops" and "grizzlies" and "under currents," in order first to break the cement by concussion, then to separate the larger-sized boulders and catch the gold in the finer material. When the deposits are very large such a method may be possible; generally, as it costs too much in time and labour, it will not be profitable. Disintegration by exposure and then washing in large revolving iron pans seems to be the most profitable method that has as yet been used. The bottom of the pan used for this purpose is perforated, and is so arranged that after a certain amount of the cement has been treated, and all the fines extracted by water and passed into the sluice, a door in the bottom of the pan may be opened to let out the material too large to pass through the perforations into the sluices, so that it may be thrown out into the tails, which collect on the ground below. In the pan four arms with ploughshare adjustments revolve to break up the cement, and make it fine enough to be carried through the bottom into the sluices. When the cement has been weathered or frozen and thawed, this work can be rapidly done. A pan 5 ft. in diameter and 2 ft. deep can treat from 40 tons to 120 tons of such cement per day, depending on the hardness of the gravel and the power available to move the pan.

The float gold which is carried off by the water in washing the gravel of the placer deposits has been the object of considerable attention. No plan for successfully saving it has as yet been devised. The system of what is called "fly-catching tables" in

Australia, has been adopted in some of the placers of the southern gold-producing counties of California. This method consists of placing across the rivers a series of weirs which occupy the whole width of the stream, so that the water shall flow over each weir in succession. They are placed 200 ft. or 300 ft. apart and are quite independent the one of the other. These weirs are constructed on piles 2 ft. to 3 ft. in length, driven into the bed of the stream, on which boards are placed which are covered with coarse gunny-cloth on the upper side. This gunny-cloth is laid down in sections more or less wide according to the size of the stream, but narrow enough to allow of the overflow of water being cut off from any one of them by means of a board held in one hand when the cloths are to be washed. The water is thus turned off from each compartment in succession by placing the board on the upper side of the stream, and the gunny-cloth with its contents is lifted and washed in a tank. Generally each cloth is washed every day. When one is taken up a fresh one is put down and the contents of that washed into the storage box which is used for that purpose only. The construction is not expensive, but these weirs are difficult to keep in repair, being subject to accidents, not only from the water but from animals who damage them and men who use them as bridges to cross the streams. In the rainy season, in a rich gold-bearing cañon, it is estimated that from $10 to $30 per week is saved in float gold on these "fly-catchers," the quantity of course depending on the richness of the washings above. It is found that when the workings above are very rich, less gold is saved than when they are poorer. Though the weirs are inexpensive they are so liable to damage and require so much watching and care that the process, though successful where it has been applied, has not been very extensively used.

Beach Mining.

It has been known from the very earliest times that gold occurred in river sands, and it is there that it is chiefly looked for in the early exploring expeditions in any country. That it existed to any extent in the sands of the sea beaches does not appear to have been known, or that sea water even contained a

very minute portion dissolved in it was not discovered until within a few years. The discovery of gold on the sea beaches of California and Oregon seems to have been, like most of the other gold discoveries, the result of an accident.* A party of miners prospecting in Northern California in 1850, of whom Eugene de Bertram was one, were forced by the jungle of the main land to follow the line of the beach. In passing along the shore at what is now called Gold Bluff, in Clamath County, they found it dazzling yellow in the sunlight, and covered with gold, which the natural action of the waves had separated from the beach sands, and so concentrated it that it yielded nearly 50 per cent. of precious metal. Collecting some of the sand, they went first to Trinidad, about thirty miles distant, and from there to San Francisco, where a company was formed to mine the beach and collect the gold. Excited by the richness of the sample, and the truthful though somewhat exaggerated story of the miners, an expedition to Gold Bluff was formed, and started from San Francisco in a revenue cutter. In order to keep the secret from the people of Trinidad they anchored off Gold Bluff, and endeavoured to land on the spot where the gold had been found. The first boat contained six persons, but on nearing the shore it was swamped, and all were drowned but De Bertram, who was rescued by the adroitness of a friendly Indian, who happened to be on the shore at the time. The rest of the party returned in the vessel to Trinidad, and went on foot to the beach. But there was no gold there, and no trace of it could be found. Discouraged and disheartened, most of the party went back to San Francisco, but De Bertram, though weary and almost hopeless, yet believing that what he had once seen he could find again, waited and watched until a favourable wind and tide washed the gold up again. The result of this discovery was that soon the beaches were explored, and have since that time been worked with moderate profit to the owners; not sufficient, however, to cause any very great excitement or the investment of any very large amount of capital. As the concentration is done without cost, and the troublesome question of tails does not have to be taken into consideration, since the sea takes care of them, the exploitation of these sands

* Min. Com. Report, 1870, p. 87.

promised to last into the distant future, and to give a rich return for the labour and capital expended. The excitement caused by their first discovery soon received a damper, for with the rude and careless methods by which they were worked, not more than one-fifth of the gold was separated, and the uncertainty of the winds and the waves bringing any up for a considerable length of time—for it was necessary that the wind should come from a certain direction to cause the light particles of gold to be brought up on the beach—soon proved that no great fortunes were to be made here. These beaches were apparently soon exhausted, and for a number of years were abandoned, on the supposition that they were no longer worth working. Many persons, supposing that these golden sands came entirely from the deep sea, undertook to dredge and to work the sands with complicated and expensive machinery, which, however, soon proved to be an entire failure.

These beach sands occur in the northern counties of California and the southern counties of Oregon, extending from Cape Mendocino to the Straits of Fuca. They are also found in small quantities as far south as San Francisco, and as far north as Puget Sound, but in too small quantities to pay for working, the district where it pays to work being confined to the above limits. The diggings extend along the coast for 300 or 400 miles. The bluffs vary from 110 ft. to 500 ft. in height. They will average 200 ft. Most of them are vertical, but they are sometimes overhanging and have very narrow beaches at their foot. High up on the beach, where the action of the waves is very gentle, the sand is found in alternating layers of the fine material composing it, with the lighter material on the top. When, on a windy day, the light sand of the upper surface becomes dry, it is often blown away to such an extent as to leave the heavy black sand on the surface. Lower down, however, where the surf breaks, the sand is coarse, and in heavy weather may be composed almost exclusively of pebbles. In California, as in New Zealand and Australia, these gold sands sometimes extend considerable distances into the ocean. Opposite Crescent City, ten miles out to sea, gold was found in the soundings, brought up by the lead, but all attempts to dredge it have failed. These sands have

doubtless been deposited by some ancient river flowing through the gold region of Oregon and California, which for ages carried in suspension, in its current, immense quantities of auriferous gravel, and which gradually filled its bed and the surrounding country with its deposits, and subsequently became elevated to their present position.

In some cases the whole length of the bluff is gold-bearing, with rich and poor streaks, yielding on the average 20 cents to the cubic yard. The rich streaks yield as much as $135 of gold, .957 fine, to the ton of sand.* The sand contains gold and platinum in small quantities, in minute scales, most of which are flat, mixed with a very large percentage of black, magnetic, and ordinary quartz beach sand. The shape of the gold grains, which can be seen when they are highly magnified, is oblong and doubly convex, with edges thicker than the middle, a form which would naturally result from the continued pounding of the grains. Their thickness is rarely more than 0.001 of a line. In length they vary from 0.3 to 0.7 line. Much of the black sand is composed of grains of nearly uniform size, but as they are not malleable they are rounded, so that although the gravity of the sand is only one-fourth that of gold, the buoyancy of the latter counteracts this, and it is very difficult to separate them. The gold is not only buoyed up by the air, but is as easily moved by the water as the black sand itself, so that it requires the greatest skill to obtain "colour" either with the shovel or the pan.

It was at first supposed that these beaches were confined exclusively to the coast line, but it has since been discovered that older beaches containing the black sand exist as much as thirty miles back of the coast line, and often at considerable elevations, and in such a position as to show that their elevation must have been very gradual. The black sand containing the gold is sometimes covered to a depth of from 3 ft. to 60 ft. with sterile material. In Oregon these are called "back beaches;" they are worked either by stripping or tunnelling.

All the early methods of extracting gold were based on the difference in specific gravity. The best of them did not save more than 20 per cent. of the gold. Plattner's process was

* Prod. Gold and Silver in the U.S. : Burchard, 1884, p. 557.

tried on the beach sands, but was found to be too expensive to be used on a large scale, but the use of it for a short time confirmed the fact that fully 80 per cent. of all the gold contained in the sands was being lost. When these sands contain gold that can be seen by the microscope, most of it is quite bright, though it is covered occasionally by a coating which prevents the contact of the mercury. In most cases this appears to be of an organic nature, and seems to have been produced subsequently to the formation of the bluff.

The remarkable purity of the gold obtained from the beaches has attracted a great deal of attention. This is undoubtedly owing to the fact that this gold, being in an extremely finely divided state, and wet with the salt water for so long a time, has had the silver separated from it in the state of chloride, as is also the case with the other metals, leaving the gold almost pure. The value of these sands is very variable, depending on the situation and the state of the natural concentration at any particular time. Values of from $10 to $30 a ton are quite common, but the sand often has less, and sometimes contains many times this amount.

As the operation of mining is entirely dependent for success on the state of the weather, the direction of the wind, and the force of tide, it is always uncertain, but as it requires very little outlay for plant, and not a great amount of labour, it is quite attractive; but as the concentration of the gold in sufficient quantities to work is dependent on such uncertain conditions as the weather, the best claims are often unworkable for months, during which time the miner must wait patiently for any change of wind which will throw the sands up on the beach where he can reach them. He may not even be rewarded for many weeks for his patience in waiting by any days of favourable winds, or even by a single tide. The sea-water cannot be used for concentrating by hand without great loss, as it is too heavy, so that the beach miner must wait for nature to do the concentrating, which, when once effected and collected for treatment, may be entirely washed away in a very few minutes by a contrary wind or a heavy tide. Such uncertainties do not invite the investment of any large amount of capital.

A favourable wind, with the heavy swells at high tide, drives the sand upon the beach. The ebbing tide, as it recedes, washes out the fine particles of rock, the lighter materials retreating with the water, leaving the heavier ones, the iron, the gold, and the platinum, high up on the beach. When the tide is out, this sand can be scraped up and "packed" inland out of the reach of the waves, to be treated there. In the intervals between two tides, or when sufficient has accumulated, the sands are washed in a sluice, terminated by a tom, with the ends and sides turned up. The bottom is perforated with holes, and underneath is the slightly inclined trough with silver-plated amalgamated copper plates. The sand passing through the holes strikes the plates; the heavy particles of gold are caught on the amalgamated plate and by the mercury, while the sands and the float gold run off at the end of the sluice The method is imperfect, all the more so as the iron-sand is more difficult to remove from the buoyant particles of the precious metal which characterise these sands, and which are very much lighter than those found in ordinary placer mining, chiefly on account of its state of very fine division. Much of it floats, and can be made to sink only by standing for a very long time, or by heating the sand, which is entirely impracticable in a large way. The probable quantity of gold is very uncertain, for a single storm will often wash out all the sand to the depth of 6 ft. or 8 ft., leaving the rocks entirely bare.

It has been supposed by some that this gold is cast up from the depths of the sea, but this theory does not seem to be a probable one, for the reasons that the attempts to dredge it in deep water have not been successful. Besides, all the bluffs which skirt these beaches contain gold, and the black sand which covers the gold is also found in a number of places several miles inland, at a height very much above the present level of the sea. There are indications that the rivers have in former times been blocked by land slides, which have sufficed in some instances to turn them from their courses into other channels, and that the rivers themselves have carried down immense deposits into the sea, where the black sand, the gold, and the platinum, in the more tranquil waters would tend to separate by gravity. There is also abundant evidence that the front of all the cliffs is gradually receding,

and as they contain gold in sufficient quantity to be worth working, this satisfactorily accounts for the presence of gold in these beaches. It is also constantly observed that after a very heavy storm, which undermines the bluff, and causes large quantities of fresh material to fall, the sands are not only richer but the gold is coarser than before. Companies have several times been formed to work the bluffs where they were rich enough to be worked. This would seem to be a very attractive enterprise, since the bank could be cheaply blasted and the sand hydraulicked, while the sea would do all the concentration and would at the same time dispose of all the tails. It has found many enthusiastic adherents, but has, however, never been extensively tried, for the distance from which fresh water would have to be brought to wash the sand, sometimes forty or fifty miles, makes the process impracticable. In many cases, especially in the interior of the country, the gold of the black sand, which is very fine, does not readily amalgamate until after it has been heated, which seems to destroy a film of organic matter with which it is coated, and which has consequently prevented the contact of the particles of gold with the mercury. It has been proposed to treat these sands by Plattner's process, but this has also been found to be generally impracticable, although the attempt to do so revealed the fact of the enormous losses that were being made by treating them by any other process.

It was very generally supposed that in the first mining the greater portion of the gold contained in the beach-sand was saved in the appliances that were used to collect them, and subsequently that the beach-sands themselves had become exhausted. But careful assays have since proved that by far the larger part of the gold was lost in all the earlier workings in the state of light films or of an impalpable powder, and in such small particles that it is scarcely visible with the microscope. How to separate these fine particles of gold from the heavy black sand has been a problem which if the sands had been richer would undoubtedly have been solved. The extreme lightness of the microscopic particles of gold, and the much greater weight and larger size of the grains of sand, make it impossible to separate them by water or air, and in many cases the gold itself is coated with some substance which prevents the

mercury attacking it. As there are millions of tons of sand that contain from $5 to $10 in gold to the ton, and are sometimes as high as $100, the experiments which were made to save it by chlorination and other chemical methods confirmed the losses which were being made in the working of the sands, but suggested no practical method of treatment.

One of the earliest-worked and most famous of these localities is at Gold Bluffs, in North California. The coast line after leaving Klamath River is very broken and rocky. The sands appear at Gold Bluffs about four miles south of the river, where for a long distance there is almost an unbroken line of cliffs varying from 100 ft. to 500 ft. in height. Some of these bluffs are vertical, and, in some cases, overhanging. They are composed of regular horizontal strata of beach-sand, gravel, and boulders. Beneath these are the trunks of very large redwood trees, lying horizontally, and stumps standing in their natural position, but below the present ocean level, showing a change of level both in the land and sea. At low water there is a narrow beach, but when the tide is full the sea washes the base of the cliffs. The method of working is to wait until the surf strikes the beach at an angle.* When the direction of the wind is such that the surf breaks perpendicularly on the beach, masses of coarse gravel are thrown up, and no black sand, but when the waves strike the beach at an angle the gravel is washed into heaps in certain places, and in others black sand, more or less rich in gold, is deposited. During heavy weather the beaches cannot be worked, as they are usually too narrow, and the action of the waves prevents the deposition of the black sand. During a severe storm the whole surface is covered, so that the surf strikes the base of the cliff, and when it subsides the shape of the beach will usually be entirely changed. Generally, the sand containing the gold lies on the top, but sometimes the black sand is covered over with a deposit of barren material, which has first to be scraped away before the black sands can be collected.

At Ophir Beach, near Rogue River, Oregon, the beach below high water is washed in a small way by the sluicing process when the sand that is thrown up is rich enough to pay. The

* Min. Com. Rept., 1874, p. 145.

superintendent examines the beaches every day, collecting the sands and scraping up the loose gravel; he pans it, and on finding a "prospect," sends for a mule train, each mule carrying two coarse hide or canvas bags, which will contain about 125 lb. each; a train of forty mules would, therefore, transport 10,000 lb., or 5 tons, at a single trip. In order to collect the sand the top gravel is stripped and the black sand gathered into piles. The canvas sacks are taken from the mules and filled, the mules being trained to start on a run for the works the moment they are loaded. As in some places the sea comes close up to the cliff, the animals are trained to turn their heads toward the cliff, bracing their feet firmly, allowing the waves to dash over them, and then going on their way on a run till the next wave comes. During a single tide these mules, under favourable conditions, are able to make three trips, when the distance to the working place is not too great. When the distance is very short and everything else is favourable, they have been able at times to collect as much as 100 tons of the sand whose average value was about $5.00.* The ordinary day's work is, however, not much over from 7 tons to 10 tons. At the works the sand is piled up and washed in rockers, long toms, or sluices, with amalgamated plates.

At Randolph, in Oregon, at the mouth of Whiskey Run, the gold in the sands was always very fine. No nuggets of more than 4 cents to 5 cents in value were found, but the sand was very rich. Only a small portion of the gold was saved. Much of the ground was worked over six or seven times in succession, and was each time found to be remunerative. For a very short time a single miner could earn $1000 a day,† but the prosperity was of short duration. The town has been in ruins for many years. The largest amount ever taken out of any of the California beach claims is said to have been $25,000 in one year. The great difficulty in working the beaches has always been the lack of fresh water. They have been washed with salt water pumped from the sea, but the amount of gold collected in such cases was only a small proportion of what was lost.

All the experience of these beach mines goes to show that

* Prod. Gold and Silver in the U.S., 1884, p. 556.

† Min. Com. Rept., 1873, p. 207.

immediately after a heavy cave or slide of the banks the beaches are rich in coarse gold, but up to the present time it has not seemed profitable to do any bank blasting or to undertake mining on a large scale by the hydraulic processes. The gold is washed up by the sea only. Many suggestions have been made that the gold would be found in the sea beyond the surf line, and some attempts to dredge for it have been made, but unsuccessfully.

It was proposed at one time to effect the amalgamation of these extremely light particles before endeavouring to concentrate them, by heating the sand and quicksilver under water up to the boiling point, which would cause the mercury to diffuse itself through the entire mass, and thus attack the gold that was in a proper condition to amalgamate. If this sand had been previously calcined, the first calcination and the subsequent heating would be sufficient in most cases to remove the coating upon the gold. In some cases a chemical treatment of the sand was made, and after all this was done, the fine particles of amalgam had still to be collected by boiling, and unless they were very rich and fuel plenty, it is evident that such a method as this, although patented and supposed to be worth a great deal of money, could not be successful. The process* consisted in screening the sand and subjecting it for twenty-four hours to a solution of caustic potash and salt, in order to remove any coating upon the gold, and to destroy the compounds of sulphur which would attack the mercury. The material so treated is heated in a pan by a jet of steam, and the quicksilver poured in; it was kept constantly in agitation by the steam, and the movement kept up from a quarter to half an hour, by which time the gold should have become thoroughly amalgamated. The pulp was discharged into a vat to cool. It is desirable that it should not be heated too much, as that divides the quicksilver more than is necessary to secure the amalgamation. Care must be taken to use pure quicksilver in the process. The cooled material was then passed over a series of copper rollers, grooved spirally and set, the one above the other. These are set over a sluice, 3 ft. wide, with amalgamated copper plates and grooved copper rolls. The rolls are hollow, 1 ft. in length and 2 in. in diameter. As the amalgam accumu-

* Min. Com. Rept., 1875, p. 317.

lates on the rollers it is either cleaned off or drops into the sluice below. Notwithstanding all that was hoped for it, the process was not successful.

It does not seem likely that beach mining will ever be very remunerative; the uncertainties connected with the value of the sands, and the times when the beaches can be worked, for they will often be unworkable for months together, either because the storms are so high as to denude the beach, or the weather so calm that there is not force enough in the waves to throw up the black sand on the beach or wash it down from the cliff, make it too hazardous. It is not probable that it will ever justify the investment of a large capital. That it will, however, yield a fair return for a small outlay, is generally true. Persons residing in the vicinity of these beaches have often made a fair living by working them when the weather allowed it, but no company, whatever its prospects at the commencement, and however rich the beach when they began to work it, has ever been able to work these sands with profit for any considerable length of time. There are places where, if fresh water in sufficient quantities could be had, the bluffs might be worked by hydraulic methods, but even this would not be justified if the line of ditch was to be very long, or if the storage of the water required the construction of large dams. The immediate disposal of the tailings by the sea looks at first sight very attractive, but it remains to be seen whether the accumulation of enormous amounts of débris would not stop the encroachments of the sea, and whether in working such fine gold on an enormous scale, sufficient of it could be caught to make the enterprise profitable.

River and Bar Mining.

Bars are the deposits that form in streams which bring down auriferous gravel, when the stream strikes a hard rock, and is deflected to the other side of its course. Parallel to the line of deflection, at an angle more or less great to the course of the stream, small bars will form, on and in which the gold lodges. In a modern stream these bars will, at low water, rise above the surface, and are then worked in a small way. The ancient rivers, however, formed bars upon a much larger scale. Such bars, run-

ning across the channels of these old streams, most of which have disappeared ages ago, can be worked with great profit. They are found where the river still retains its ancient course, but the diminished size of the modern stream will frequently make these bars from 40 ft. to 50 ft., or even higher, above the present high-water level. They are ordinarily worked by the hydraulic methods.

In very many rivers in California and elsewhere which run through alluvial deposits containing gold, or which have abraded the outcrops of veins containing it, there is a deposit more or less large of gold in their beds. In this case, which is very common in California, four methods are used. One is to dredge or pump

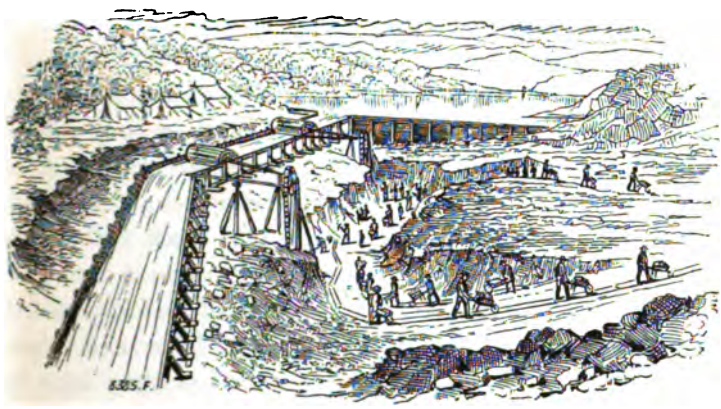


FIG. 6. RIVER MINING.

the sand from the river bed, or to undermine it. A second, when only a very small portion of the river bed is to be removed, is to sink a cofferdam. This is a method now principally used by the Chinese. A third is to remove the whole of the water from the river by means of a flume and "flutter wheels," Fig. 6, and then to work the ground of the river bed and beneath it, returning the water to the river at the end of the flume. As many men as can possibly work together are put in the dry bed. They work day and night without interruption until the rise in the river warns them to take out the wheels and pumps, and either abandon the place or wait until the next season. This method is still in use on the Salmon, Scott, and Klamath rivers

in the north-western part of California. Such experiments as these are extremely hazardous, as the least defect in the construction of a flume or a freshet may break up the whole of the works, the outlay for which must of necessity be very large. They can, therefore, only be undertaken during the summer, when the water is low and when there is no danger of sudden rises in the river. Flumes commenced before the spring rains were over have more than once been carried away before they were finished. Usually these river claims, as they are called, are kept clear of the water coming in from leaks in the dams, and from the water of small streams which enter the sides by undershot wheels which work in the flumes constructed to carry off the water by the force of the current. Still a fourth method is to cut off the flow of water in the river by turning the whole of the stream into another channel.

Where the bars are simply in the rivers of the present time they are of local and transient interest only, as they are very soon worked out. The gold they contain has sometimes been so concentrated by the action of the stream that a single panful has been known to yield more than $300. Such rich pay was generally worked over two or three times before it ceased to be profitable. Such claims have been pumped and dredged, but they do not generally justify any large expenditure, and all workings based on any large yield for a period of time have been unsuccessful. Such plans were proposed at Oroville on Feather River, California, but the failure for most of them, owing to the precipitous nature of the banks and the rapidity of the current, was as great and disastrous as the expectations were large and promising. A few of these claims made very large profits, but as no work could be done on Feather River until July or August, it frequently happened that either the flume and dam were washed out, by a sudden rise of the water before the work of mining proper had commenced, or the time they could work was so short that sometimes the cost even of the work was not realised. Occasionally, where wing dams were put up, it was found that either the dam or the sluice rested on the richest spots and crevices, and these of course could not be worked at all. The value of any individual portion depends very much on the

nature of the bedrock. If its outcrop crosses the stream at such an angle as to form natural riffles, the gold will lodge there and the spot will be very rich. If, however, the rock is flat it will only be caught in the sands and in the crevices between the cobble stones, and most of the gold will be carried great distances until it finds a place where it can lodge, and is thus distributed in a very large amount of sand. Below Oregon bar the river was found to be very rich, but it was so difficult to keep the water out of the workings, that it was worked over three times by white men and twice by Chinamen. The Chinese are, however, working many of these deposits and make a fair living out of these abandoned bars or their tails. From 1849 to 1883 the Middle Fork of the American River was the seat of great activity in river and bar mining. The products of these claims would today be considered fabulous. Those who could not get claims on the river followed up the gulches until the source of the river gold was discovered 2000 ft. above the bed of the river at the line of contact between the bedrock and the overlying cap of volcanic ash.

Feather River,* in Butte County, California, was one of the most famous for its rich diggings. On the "Cape claims" two wing dams were constructed across the river. All the water above the first dam was carried by a large flume to the lower one. The water between the two dams was pumped out by wheels run by the current in the flume. The gravel on the river bed was washed in sluices, and during forty-two working days yielded $680,000 from 330 linear feet of river bed. Only one-third of the ground had then been washed when the flume was destroyed. On the Rock Island claim in 1854, there were only two working days after the dam and sluices were finished, but Chinamen took out $30,000 in thirty hours. In 1855 the river was again flumed at the same place, but before the breaking of the dam, by the high water, the long toms were yielding $1000 an hour. On one claim where they worked only three days they took out $15,000. One company who finally got down to the bedrock took out $40,000 in twenty-four hours. At the Jones claim, above White Rock,

* Prod. Gold and Silver in the U.S., 1880, p. 328.

one Fall, when the river was rising and began to pour over the dam, the men rushed into the water to gather the sands; some of them got $80, some $200, and it is said that one man who, at the peril of his life, stood until the water was up to his shoulders, stooping down to pick up the sands and washing it as he stood, gathered $600. It seems wonderful that such riches should not sooner have attracted attention, but the danger and uncertainty of working was such that, for individual efforts, the risk was very great, while the profit was uncertain and comparatively small.

Such methods can only be employed by individuals, on small streams in the dry season, when there is the least water to contend with. The stream must be diverted from its course and pumped out. Such methods, as shown by the examples cited above, are full of risk. They seem very attractive when the rich pay dirt is brought up from the bottom and the calculation is made of how much gold is contained in the river bed, but it is seldom worth while to flume them for a short distance—generally it will not pay to do it for less than 300 ft. or 400 ft., and often more must be flumed—and when all the water has been pumped out, derricks run by hurdy-gurdy wheels or by hand power must be erected to remove the large rocks to get at the gold beneath them, with no certainty that the whole of the work will not be swept away by a sudden rise in the river.

When such a method is not practicable the river is explored to find the bars and rich placers. When they are located, a shaft is sunk to the depth of 40 ft. to 60 ft. above high-water level, and a drift run in the rock, or in the sand, under the bars. The gravel is taken out from above, hoisted to the surface, and washed in sluices. The water pumped from the shaft can, if necessary, be stored for washing. Gravel containing only 70 cents a cubic yard is worked in this way.

Where the quantity of water is, or is likely to be, small, in order to economise it, it is the custom to dam the stream, and, when the water has accumulated sufficiently, to open the gates and allow it to rush out against the gravel with great force. This process, which was well known to the ancients, is called "booming;" it is sometimes carried on by hand power and sometimes automatically. When the process of "booming" is carried

on automatically it is effected with the use of the so-called "shot-gun dam," or the "self-shooter," as it is called in California. This is an automatic discharge gate, and is in California so constructed as to have the gate of the dam lifted at intervals by the water itself; to do this the gate is attached to one end of a suspended beam, which has a water-tight box with a valve in the bottom, attached to the other. When the dam is full a part of the water flowing over passes into a launder, arranged for the purpose, which conducts the water to the box. This is so adjusted that when it is full it lifts the gate by itself sinking. The contents of the reservoir is thus instantly released, and flows with great force and rapidity to the point where it is to be used. After it has fallen a certain distance the valve of the water box comes in contact with an upright post, which opens it and lets the water out. The empty box rises and closes the gate. This is repeated at regular intervals, giving a heavy head of water and a rapid current every few minutes. In Colorado the gate is made of a heavy piece of canvas. To this a rope is attached which passes up over a 5 ft. pulley and at a distance of 15 ft. over a much larger pulley. To the end of the rope which falls from this pulley, the water-tight box is attached. The rest of the construction is the same as in California. This method has been extensively used both in Colorado and California. Before the automatic appliance was invented, the gate was always lifted by hand and required the constant attendance and care of one man. By the present method the gate requires to be looked after only occasionally. The work is entirely automatic, the number of times per hour that the gate is raised being perfectly regular, depending only on the supply of the water. It is used when water is scarce and labour not easily had.

It has been proposed, and a great many attempts have been made, to raise the gold contained in the river gravels from the bottom of the streams with powerful pumping engines, capable of raising 25,000 cubic metres in the course of twenty-four hours, half of the bulk being water. It is claimed that these pumps are cheaper and better than any dredging machines. Notwithstanding these assertions, however, neither dredging nor pumping has proved successful in this country, for the reason that the river

can only separate the gold in the same way that a sluice does. The idea of most persons not familiar with river sands is that the sand is deposited in regular stratified layers, and that the gold is disseminated regularly through them. No one who has any experience in sluice mining would believe this, for a river is only a natural sluice. The whole object of the sluice is to provide places where the gold may be caught in mercury, the bottom of the sluice being arranged with reference to this. At the bottom of the river the sands are always unequally distributed. The bedrock is never even, but is always filled with crevices, into which the gold falls by gravity and is not accessible to the dredge. If the rock is moderately hard, and the strata is very much inclined, the rock will be certain to be broken off unevenly, forming irregular depressions and projections. If the course of the stream is at an angle to the strike of the rock, then every projecting piece will serve as a riffle across the stream behind which the gold will lodge. If the direction of the stream is the same as the strike of the rock then the inequalities of the rock will form long and irregular longitudinal riffles in which the gold will collect and hide itself. Every irregularity or inequality will be a place of deposit for the gold. Where the river flows over a soft gravel, and the bedrock is soft from decay, it is well known to gravel miners that the gold will penetrate the bedrock sometimes for a considerable distance, so that it is frequently worth while to take it up for some depth. Dredging and pumping machinery must do their work progressively and systematically, starting from a fixed point and proceeding toward some other. It is impossible for the machine to find where the gold lies, and even if every spot were known beforehand, the irregular nature of the deposit, and the fact that the bedrock is often the richest part of it, would make it impossible to work a spot here and there, so that in pumping or dredging the barren or poor material will be more likely to be raised, even supposing the gold is actually there, than that which contains the precious metal in any large quantity, as the gold is hidden away in pockets beneath ground, which cannot be reached by the dredge. These processes, therefore, have not been successful in this country. They have also failed in Brazil, where it has been

tried on a large scale. It is said that it has been successful in the Otago district of New Zealand. It may be stated as a general proposition that if the river bed is poor it will not pay to work it by any method, and if it is rich the usual process of damming and fluming, which with all its uncertainties has sometimes proved successful, is probably the best, as it is certainly the cheapest method of working, in dry seasons, when the flumes are properly constructed. In such a case the practised eye of the miner distinguishes the points where the gold is likely to be found, and he works those points only, leaving the barren material behind; but the machine must take poor and rich as it comes, and cannot in any case work the bedrock. It has been stated that dredging on a small scale has been tried on the Carson River, which received the tails of the Comstock, and that material ranging in value from $5 to $75 per ton, of which 50 per cent. is mercury and amalgam and 25 per cent. sulphurets, which has settled from the tails, has been raised. It is very doubtful whether such an enterprise could succeed even if there were large amounts of tails.

Efforts are now being made to revive river mining, and some work is being done in driving tunnels to divert the waters of the streams entirely from their course, so as to leave the beds of the rivers for long distances altogether bare, so that there will be no risk of the water coming back on the beds to disturb the work. It is a much better method of working, but requires such an outlay that no private parties can engage in it. If these projects now in course of being carried out are successful, it will probably revive the working of river beds. The method of diverting the streams and rivers into other channels by means of tunnels could hardly have been possible until within a very few years. The cost of driving tunnels and the immense amount of time required to execute the work led to the failure of more than one enterprise in the case of the drift mines, some of which took as long as six, eight, or even ten years to open, and have then been abandoned from weariness of waiting for results. One of the most notable instances on record of the execution of a long tunnel which could never have been undertaken in former times on account of the time required to execute it, is the Big Bend Tunnel, of Feather River, in Butte County, California, whose object is to

divert the North Fork into the West Branch of Feather River, and so lay bare the entire channel of the Big Bend, which is over fourteen miles long, so that the sand in its bottom can be washed for gold. It has been known for a long time that the gravels of this river at this point were rich in gold. In fact, the discovery of gold on Feather River was made very early and led to a most careful prospecting of the length of its entire bank. Great value, as has been already remarked, had been taken from its bed. Portions of the river were, however, on account of their very rapid currents and steep banks, inaccessible to the miner, even after their richness became known. All of the bed of this river, at this point, has paid whenever it has been made accessible. Generally it has been impossible to work more than a few feet of the banks, rarely reaching the bottom of the river bed, but when the bed-rock was reached, it has always been found to be rich. On one occasion $3000 were taken from a crevice near the edge of the water. Values of from $15 to $20 the rocker were common. The almost semicircular curve in Butte County known as Big Bend, shown in Fig. 8, was an instance of this. Occasional washings of its bars and hurried workings made in the gravel of its bed had shown great richness from time to time. The rocky cañon through which the water flows for fourteen miles through the Big Bend is very wild, and accessible only with very great difficulty. The channel of the river is at the bottom of the cañon, which is from 1200 ft. to 1500 ft. deep. The precipitous nature of its sides, and the difficulty of working the river bed, is shown by Fig. 7, which is a view at Island Bar in the Big Bend. Explorations above and below the Big Bend had proved it to be exceedingly rich, and large quantities of gold had been taken from the bed of the river. The nature of the ground had prevented working the river bed, as neither wing dams nor flumes could be constructed in it, excepting in a very few places. The river itself has such a fall as to create a strong current, and the volume of its waters is rarely less than 80,000 miners' inches. The miners of the region had proposed to attack the problem of draining these fourteen miles of the Big Bend,* when

* I am indebted to Mr. R. V. Pierce, of Buffalo, New York, for most of the information relating to the Big Bend Tunnel.

a company formed for the purpose began work in July, 1882, the object of which was to drive a tunnel large enough to carry the whole flow of the river, which, on account of its steep banks, it was impossible to divert from the bed in any other way. This seemed to be justified, as profitable workings had been carried out both above and below the Big Bend. One of these was at Island Bar, Fig. 7, where wing dams had been constructed and the water flumed, but it had only been partially worked. All of

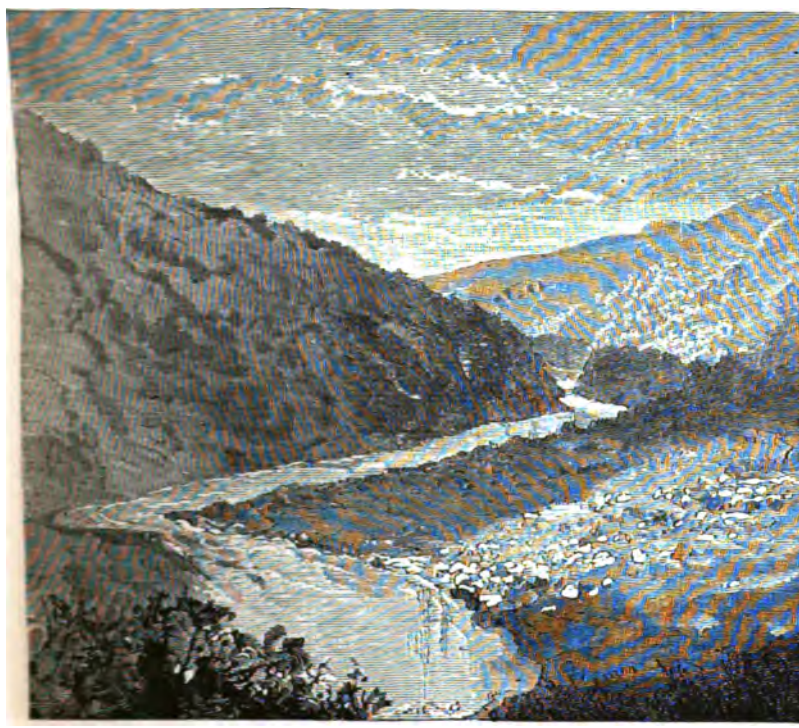


FIG. 7. VIEW OF ISLAND BAR, FEATHER RIVER.

them had proved to be exceptionally rich. The river at this point runs through a narrow cañon 15,000 ft. to 2000 ft. deep with very steep and abrupt banks, which gives very few facilities for working, and consequently made this method of mining extra hazardous. During the summer of 1882 very careful surveys of the whole region were made, as the result of which it was ascertained that a tunnel 12,007 ft. long, or nearly two and one-third miles,

with an average grade of 32.1 ft. to the mile, would carry the waters of the river from above the Big Bend to Dark Cañon, from which they could be made to pass to the West Branch and eventually reach the main river at a point some distance below the bend. It is proposed to use this water principally for irrigation, which will make the region three or four miles from the outlet agriculturally very rich. The water can be easily carried by pipes and ditches at easy grades to fine fruit and other agri-

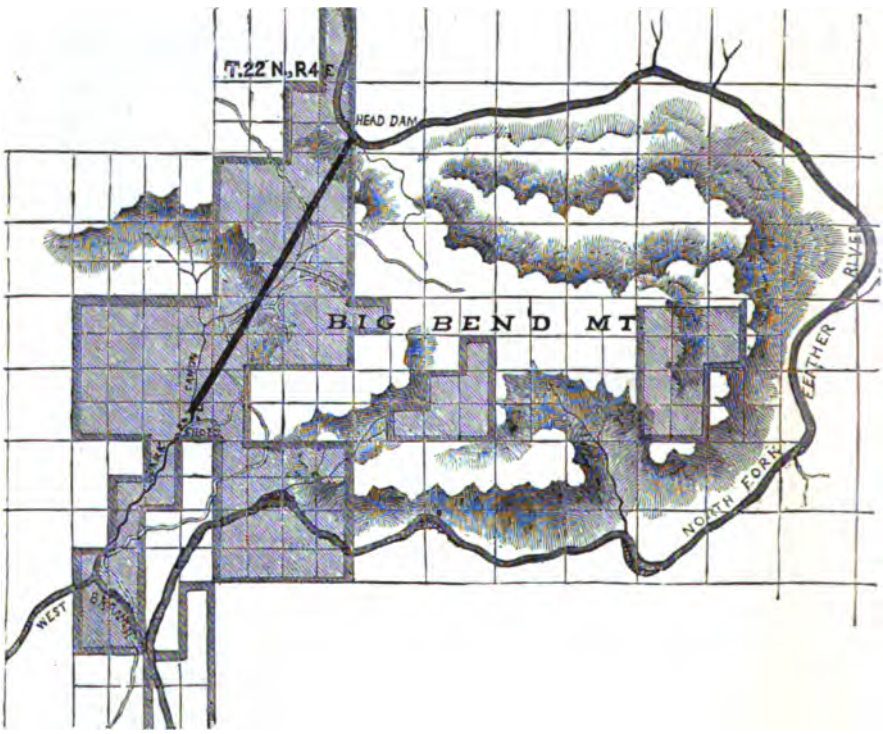


FIG. 8. MAP OF BIG BEND AND SECTION OF THE LINE OF THE TUNNEL.

cultural lands. Some of its power will, however, probably be used for manufacturing purposes. Besides baring the river bed, the turning of the water will make it possible to drift in and bottom the deep gravels of Oroville, which the proximity of the river and the nature of the ground has not yet admitted.

It will be seen on Fig. 8 that the shape of the river at this point is like a horseshoe whose ends are only three miles apart. Near one extremity, Figs. 8 and 9, Dark Cañon, a narrow gorge

running nearly north and south, and ending in the West Branch of Feather River, comes in. The bottom of this gorge is many feet below the bed of Feather River at the commencement of the bend. By diverting the waters in this way the entire bed of the river for fourteen miles would be exposed to mining operations. The map, Fig. 8, shows the position of the river, the dark lines being the property, in addition to the river banks, which is owned by the company. Fig. 9 shows the longitudinal section of the tunnel. It was decided to run the tunnel an upward grade of 29.7 ft. to the mile until 300 ft. from the upper end, from which point all of the unused grade will be utilised in giving a high velocity to the inflowing waters. The work proper, in Dark Cañon, began on November 18th, 1882. Two days later a night shift was put on, and during the following month three shifts of eight hours were established. When the work commenced, the plant consisted of a No. 4 Burley air compressor, arranged to be driven by steam or water power, an air receiver, 4 ft. by 16 ft.; a No. 3 Knowles

FIG. 9. LONGITUDINAL SECTION OF BIG BEND TUNNEL.

pump, and a Llewellyn heater, 2 ft. by 8 ft.; an 8 ft. Knight water-wheel; a buffalo drill carriage, mounting four drills; and a tubular boiler, 5 ft. by 16 ft. Four Burley drills and a No. 4 Clayton duplex air compressor, a No. 5 Baker blower, and an engine to run the blower, were afterwards added to the plant. The water for the wheel is supplied from a ditch which carries 100 miners' inches, taken from Dark Cañon; an 11-in. pipe, having a vertical fall of 275 ft., carries the water to the wheel.

The progress of the work since the beginning is shown in the following Table:*

	ft.
Distance by hand prior to Nov. 18, 1882	26
,, ,, drill to Dec. 31, 1882	373
,, ,, ,, Jan. 1, 1884	3,503
,, ,, ,, Jan. 1, 1885	3,090
,, ,, ,, Jan. 1, 1886	3,855
Total to Jan. 1, 1886	10,847
,, length of tunnel	12,007
Remaining distance...	1,160

* Scientific American, February 6th, 1886.

No full record of work was kept prior to January 1, 1883, but since then it is complete. In 1883, six days' time, or eighteen shifts, were lost; in 1884, four and two-thirds days, or fourteen shifts, were lost; and in 1885, only three and two-thirds days, or eleven shifts. This represents all time lost by reason of breakage of machinery, cleaning boilers, and all other causes; seventy-five men, working eight-hour shifts, were employed in this work. The ventilation of the tunnel was effected by the air drills and the Baker blower, the exhaust from the drills furnishing all the air required for ventilation when the men are at work. At first it was proposed to keep down the powder smoke in the tunnel by a spray of water, but the men found that the moment they began to pick up the rock thrown down by the blast, the smoke rose again, so that it was necessary to locate an exhaust at the mouth of the tunnel and drive it by means of an engine which was used for extracting the smoke and foul air. It was generally put in operation ten to fifteen minutes before a blast. At the same time the air compressor delivered a volume of fresh air directly into the face of the work. This arrangement allowed the men to resume work about fifteen minutes after the blast. Both the blower and compressor are kept at work until the débris has been removed and the drilling commenced, when the blower is shut down until just before another blast. In prosecuting the work, 154,815 lb. of No. 1 giant powder were used, and 16,535 lb. of No. 2, or a total of 171,350 lb. of giant powder. The smallest distance made in any month was in August, 1883, when only 175 ft. were driven. The greatest distance made in the same time was in September, 1885, when the advance was 405 ft. The monthly average for 1883 was 291.9 ft., and for 1885, 327.2 ft. The construction of the tunnel took seventy-five men working day and night seven days of the week, three years and four months.

The change in the character of the rock during the progress of the work makes the results of the different months, as given above, not strictly comparable with each other. During the first nine months, slate, easily penetrated, with occasional stringers of quartz and granite, prevailed. During this time only 200 ft. of a hard diorite was met. For several months after this the rock continued hard and difficult to work, but when the tunnel had

been driven about 6000 ft., or just half of its length, a black slate was met, which, though close and hard and requiring a large amount of explosive to blast it, allowed of rapid speed with the drills. Only five places in the whole length of the tunnel had to be secured by masonry. In these the top is roofed with timber or iron, and the sides and bottom cemented. The temperature did not rise above 60 deg. Fahr. during the construction of the tunnel. In several cases beds or veins of rock were passed through which yielded $8 to $14 per ton in gold and silver, and these may at some future time be profitably worked. The water was gauged at the time the river was low, and the size of the tunnel was determined at this time. It was originally 16 ft. high and 10 ft. wide, giving a cross sectional area of 160 square feet, or 23,040 square inches. The exact length of the tunnel is 12,007 ft. The headings of the tunnel met April 12th, 1886. The water was turned into it in July, 1886. To effect this a permanent dam, 200 ft. long and 16 ft. high, was built of very large timbers bolted to the bed-rock and fastened together. These were filled with rock and planked over. The inlet to the tunnel is controlled by six heavy gates 4½ ft. by 8 ft. in the clear, constructed of iron and steel and worked by screws and handwheels. The entrance to the tunnel is widened for some distance to 32 ft., and will carry 105,000 miners' inches of water. As the water rises sometimes 50 ft. in the river, these gates will be closed to shut off the water when the tunnel can no longer carry it. The river will then be allowed to flow in its own channel. When the river was turned into the tunnel, 10 ft. by 12 ft., it was found to be only large enough to carry the stream at low water. A flume had at once to be constructed to carry off the excess in any other season, so that the river bed could be worked. This flume is 18 ft. wide, 3 ft. deep. It is calculated to carry the 10,000 miners' inches or 200 cubic feet of water per second, which would otherwise be discharged over the dam. The entire volume of the river was calculated as 100,000 miners' inches, or 20,000 cubic feet of water per second. The enlarged tunnel, 12 ft. by 16 ft., will be capable of carrying all the water during the whole of the dry season, with ample allowance for any increase in volume caused by heavy Fall showers which are liable to occur before

the rainy season sets. The flume is 1007 ft. long. For this distance the river bed is bare and is being washed. Beyond this point, however, the water spreads out over the greater part of the river bed, making prospecting somewhat difficult, but the sands yielded well. When the bedrock, beyond the limit of the sluice, was reached it was found to be rich. The presence of so much water rendered it, however, difficult to reach it except in a very few cases. The work of enlarging the tunnel to 16 ft. by 12 ft., so as to make it capable of carrying 120,000 miners' inches of water, which will empty the channel completely, and keep the river bed bare from seven to nine months of the year, so that local drainage only will have to be provided for, was commenced just as soon as it was ascertained that the tunnel was too small. In order to diminish as much as possible the labour and expense of this enlargement the head of water available was made use of to remove the broken rock from the tunnel. Only the large pieces have to be loaded on cars to be removed. The larger part of the small rock will be driven out by the very rapid current when the gates are opened to let the water through the enlarged section. A track of 2 ft. gauge, laid with 16 lb. T rail, extends through the tunnel from the heading. The grade being uniformly down, the removal of the rock is not difficult. The movement of the cars during the construction was effected entirely by mules, six animals being kept at the tunnel for this purpose. The trains were composed of from ten to twelve cars. The number of daily trips is regulated entirely by circumstances. From an elaborate series of measurements made at the site of its upper end, it is calculated that the present tunnel will suffice to carry off the waters of the river for a period of from seven to nine months in each year. During the rest of the year the gates will have to be closed, so that the stream will follow its natural banks. At the entrance of the tunnel the river is narrow and enclosed by steep banks, so that it forms a favourable site for a dam to turn the water into the tunnel when the gates are open, or allow the river to flow over it when they are closed.

In addition to the main part of the enterprise, that of driving the tunnel, an immense amount of work has been necessary on the surface. Roads have been built in order to facilitate the

transportation of supplies from Oroville, some sixteen miles distant, and have been extended over different parts of Big Bend Mountain, so that timber can be conveniently brought to the steam sawmill. Fourteen miles of pack animal trail have been built around the Big Bend, in order to make all portions of the claim accessible. A private telephone wire has been built to Oroville, sixteen miles distant, as well as around the Bend, and in time will be extended to all parts of the trail. The company has purchased four thousand acres of land, in order to cover its tunnel site, provide ample timber reserves, and in order to protect it against actions for damages arising out of the backing up of the water above the proposed dam, or out of the increased volume which, as soon as the tunnel is completed, will find an outlet through Dark Cañon and the West Branch. The right to the banks and river bed for a mile above and half a mile below have also been secured.

The working of the river bed is done by long toms and sluices, 16 in. by 18 in., the water being supplied by streams on the river bank, which can be controlled but which give a plenteous supply. Between the mouth of the tunnel and its outlet in Dark Cañon Rock, Cheno, French, and Berry Creeks empty into the river. The water of all these streams is to be utilised in mining the river bed. But for their emptying into the river all the water used in washing the sands would have to be brought in a sluice from the dam. These streams enter the river at such points as to distribute the water where it is wanted. They will be used not only for working the sands but for handling the river bed boulders which have to be moved. The sands will also be hoisted by the same process into sluices set at a much higher angle than would otherwise be possible, which will allow of washing very large quantities of sand. The supply is sufficiently large to furnish all the power required in mining, and the water can be easily distributed so as not to be in the way of the miners. A large number of places will be worked at a time, fifty to a hundred men being employed at each place. In most places there is not more than 4 ft. of gravel on the bedrock, the channel of the river being from 100 ft. to 200 ft. wide, so that it can be easily worked.

Sometimes there is difficulty in working river beds and bars on account of the flat character of the country, and the

getting of such a grade as is necessary for sluicing. This is being overcome, where a sufficient head of water can be had, by forcing the gravel up a steep incline and from there working in sluices as usual.* Bar mining has ceased to pay remunerative wages to white men, and has been abandoned to the Chinese, who were for years engaged in working the banks and bars of the Yuba River, washing the ground that had already been worked over three or four times. Their system consists in turning the stream and running the dirt through sluices. They realise from $1.50 to $2 per day, and occasionally find ground that, having been overlooked, has never been washed before, and is quite rich, but this is very rare. Partly owing to the great influx of impoverished miners to the diggings, and partly owing to the fact that the shallow placers were quickly worked out, the whole industry ceased, years ago, to be very remunerative. Placer mining might have become entirely extinct in California had it not been for the discovery of deep but poor beds extending over vast areas and of so permanent a nature as to warrant the expenditure of years of time and vast capital to work them. With the decay of the shallow placer industry the miner, as he was known in the early history of the production of gold in California, has disappeared. The undaunted, self-reliant man who recognised in a failure only the incentive to try again, whom no discouragement deprived of his independence, has gone, not to reappear. In his place there is working to-day a labourer whose wages are his only reliance, and the hope that he may some day own the whole or part of a claim, his only expectation in the future. The capitalist has absorbed the miner, and engineering skill has replaced his unflinching but more or less misdirected industry. Still, the miner of to-day would rather earn but little, working for himself, than high wages working for another; but the accumulation of capital and the control of the ground by large companies, who own all the water and mineral privileges and rights, and frequently own the ground, have prevented much chance for the wage-earner of to-day becoming an independent miner.

It would be impossible to state any yield for the placer deposits as an average. Those which have been published were

* See Chapter II.

interesting at the time because they were extraordinarily large, while the poor yields have not been made known. The few successes have been heralded all over the world, but the great army of unfortunate ones has not been heard from, and scarcely excites any commiseration or sympathy. To average the few successes of the fortunate with the immense number of failures would bring the gains down so very low that few thinking men, if they were to consider the proposition by itself, would care to run the risks of placer mining; but every one hopes that he will be one of the fortunate ones, and hence many make the attempt. It is doubtful whether, over a large and moderately rich district averaging successes and failures, the yield has ever been as much as $10 a day for each man, and this, with the great cost of living in the places and times when such mining is possible, is scarcely more than a bare living. How many men never found enough to keep their hearts up, and died, either in the field or the insane asylum, from grief over the loss of the whole of the small capital they had saved to bring them to the "diggings," or from disappointment at never having realised their dreams of wealth, we shall never know, but the number must have been very large to bring the average down so low. James Marshall's life and death are an excellent illustration of the proverb: "He that hasteth to be rich hath an evil eye, and considereth not that poverty shall come upon him."*

Formation of Placer Deposits.

The origin of gold both in placer deposits and in veins, and especially the origin of nuggets, has been the subject of repeated discussions and investigations which have often been brought to my attention, by extremely interesting specimens given to me for examination, and more especially by the one exhibited at the meeting of the Institute of Mining Engineers in February, 1880.† In the year 1874, while making some examinations of the hydraulic mines of California, I was very much struck with the distribution of the gold throughout these deep placers, which were almost invariably poor on the surface, while gradually growing

* Proverbs, xxviii., 22.

† Trans. Am. Inst. of Min. Eng., vol. viii., p. 451.

richer towards the bedrock. The constant presence of fossil wood and the large quantity of organic matter contained even low down in these beds was also remarkable. Not being satisfied with the various theories advanced to account for the formation of these deposits, I began an investigation early in the year 1879 on the conditions of solubility of gold and the causes of the loss in working gold ore in a large way. These researches showed that gold must be considered a soluble rather than an insoluble metal, and that the conditions of solution are such as will be found anywhere where gold is likely to occur, and that the solution may take place even under the ordinary circumstances of surface drainage, and may be going on freely even where the presence of gold has never been suspected, and that there are causes enough in nature, to produce the solution of the gold in sufficient quantities, to account for all the phenomena of both the vein and placer formations.

The general theory with regard to the formation of these placer deposits and nuggets has been that they were the result of the destruction of pre-existing vein matter, which does not accord with all the facts as shown in the deep placer deposits. The gold in such case would be distributed in layers of unequal richness throughout the bed, the richness depending on the amount of deposition taking place at any one time, and would not occur in increasing richness from the top to the bottom. Further, every particle of gold of whatever size would have a rounded form, resulting from its abrasion against the harder rocks, which is not the case, the small as well as the large grains being of very irregular shape. It must also be borne in mind that most of the veins from which the gold is supposed to have come had a gangue of quartz. The gold is much softer than the rock; the amount of precious metal contained in the vein would also be very much less than the quantity of rock, so that in the destruction of the formations there would be a very small amount of gold being abraded and ground in a very large quantity of rock. It is therefore likely that the coarse particles of the gold, which is so much softer, would be comminuted at least as fine as the rock, and the smaller ones much finer, so that the difference in density would hardly tend to make a concentration by any subsequent

action of wind or water, since the small particles of gold would be liable to float away and thus prevent the concentration. Where the large particles are not in sufficient quantity to make an extended natural concentration possible, and where the deposition of the sediment of the rivers is taking place, the result would be a very small quantity of almost impalpably fine gold, distributed uniformly in a very large amount of comminuted rock, or a production of clay resembling that used to make brick around Philadelphia, which contains very small amounts of gold uniformly distributed through it. The structure, too, of each one of these particles would be the same as that of the rock with which it was abraded, and would be uniform. It is, however, well known that the grains of gold found in the placers are not of uniform shape; some of them are flattened with rounded edges, others rounded, and most are mammillary, all of which forms are not probable, and hardly possible, under the conditions suggested. A nugget rounded like a water-worn pebble is a great exception in any of the placer deposits.

While the theory of vein destruction might in some cases account for the presence of gold in small quantities throughout the sands in grains large enough to admit of concentration, it could never account for the presence of large nuggets which, if they had been transported any distance by water, would have lost their mammillary form. Admitting the greater size and force of the ancient rivers, it is impossible to conceive that such large and irregularly shaped nuggets as those from Australia, Siberia, and from this country, could ever have been so transported by water as to be entirely relieved of all their gangue, without having themselves assumed much more regular surfaces and a more uniformly cobble-stone shape. Besides this, the sharp, or comparatively sharp, angles of the sand show that there has not been any great amount of transportation by water. On the other hand, slow accumulations from solutions of varying strength and a deposition of unequal rapidity continued for a great length of time accounts perfectly both for the form and for all the attendant phenomena. It is a fact, moreover, that very large masses, such as these nuggets, have never been found in veins, and are confined exclusively to placer deposits. The detrital theory accounts still less for the fact that

in many of the deposits, especially where the bedrock is soft and porous, the gold often enters it to the depth of nearly one foot, and it is frequently the richest part of the deposit.

In 1867 Mr. Wilkinson, of Australia, made a series of researches with reference to the effect of organic material on the deposition of gold. Sonstadt* also made a series of researches on the presence of gold in sea-water, and found it to be present in the ratio of about one grain to the ton of water, or about $1 for every 25 tons of water.

Up to this time gold had always been considered as a very insoluble substance, because it was insoluble, or very nearly so, in most mineral acids. Ingenious metallurgical processes, based on this insolubility, have been invented, and are still in constant use; but it does not follow that because gold is not affected by the ordinary acids it is therefore not soluble in other substances much more likely to be found in nature. The action of organic acids and of the alkalies were left out of view, and also the fact that the solution of infinitesimal quantities may acquire great significance in a geological era or even in a much more limited time.

Bischoff found that sulphide of gold is slightly soluble in meteoric waters, and much more soluble in a saturated solution of sulphuretted hydrogen in water. It has also been ascertained that chloride of gold in minute quantity in an alcoholic solution may remain in solution in the presence of proto-salts of iron, and that metallic gold is slightly soluble in solutions of the per-salts of iron. But the theories founded on these discoveries supposed that gold was much less soluble than it really is, and that the solution required peculiar agencies and a set of circumstances not likely to occur everywhere. Its diffusion in sea-water was accounted for by the presence of chlorine, iodine, bromine, and of alkalies, and these conditions were not thought to be of general application in the explanation of the phenomena exhibited in mineral veins.

In 1884, Dr. M. C. Selwyn, then the Government geologist of Victoria, proposed a theory of solution to account for the formation of nuggets and placer deposits, suggesting that the gold was dissolved by the waters which filtered through the soil

* Chem. News, vol. xxvi., p. 159; Am. Chemist, vol. iii., p. 206.

was carried in solution until it met some nucleus around which it could deposit, and was then precipitated, and that nuggets and placer deposits were formed in this way. The cause of the solution, he suggests,* was the percolation through the gold-bearing strata of very large quantities of saline and acid thermal waters, during the periods of great volcanic activity, which produced the basalts. This action accompanied, but to a great extent succeeded, the phenomena which produced the present placer deposits. This gold from meteoric waters deposited on that already in the sands, produced the nuggets. He further states that his opinion is confirmed by the fact that large nuggets only are found in the western gold-fields, as at Ballarat, Daisy Hill, &c., where immense basaltic eruptions had taken place all over the district. In the eastern and northern districts, as Gippeland, Tavra, Ovens, &c., where streams of basalt occur only to a very limited extent, or are altogether absent, the gold is generally very fine, and nuggets over 1 oz. in weight are of the greatest rarity.

These researches and theories, however, did not attract very much attention, and the old theory of the destruction of pre-existing veins was still adhered to. It is to be observed, however, that when gold does come from the destruction of veins the surfaces are rounded and worn smooth, as is shown in the large boulder of quartz containing gold detached from a vein in Venezuela, which is now in the collection of the School of Mines. This is in entire contradiction to the mammillary structure of the nuggets. If they had been transported far by water they would have been rounded and water-worn to much more regular surfaces. These worn surfaces would, of course, have been confined entirely to the outside of the nuggets, any cavities existing in the interior of the piece would have been in the condition in which they left the vein, and the edges of any crystals found there would have been sharp; while in the nuggets the mammillary form exists even in the cavities of the interior, and even where crystals or the commencement of crystallisation is observed, the edges of the crystals are very often blunted or rounded, showing both deposition and solution on these edges.

It is also to be noticed that the analyses of nearly all the

* Letter from Dr. Selwyn, dated March 28th, 1882.

samples of gold taken from veins show it to be much less pure than the nuggets found in the placer mines of the same district. If the gold of the placers had come from eroded rocks it would be of the same composition as that of the veins of the district in which it was found. It is well known that most of the gold nuggets are pure, while the gold of the veins is of a much lower grade, containing considerable quantities of silver and other foreign metals. Thus, the California nuggets vary from 800 to 980 fine,* the Ballarat nuggets are 992.5, the Australian nuggets vary from 960 to 966; those from veins in California from 730 to 860; in Transylvania 600 with 399 of silver, and in Nevada there are some of 554 of gold and 429 of silver, and others only 333 of gold with 666 of silver.

It must be remembered also that the violence of the old placer currents was very much greater than that of the ordinary streams of these days. The rivers were not only larger and deeper, but more rapid, and the results of their action would have been an almost complete comminution of the gold by its rubbing against the harder rocks. If this were the whole of the process and no further action had taken place, the gold would be found in the sands in this comminuted condition exclusively, and few, if any, of the particles would have escaped the battering and pounding process incident to long exposure to rolling rocks, and the deposits resulting from it would be found on the bed of the stream only, or a very short distance below it, where it could sink by gravity.

Gold is, however, also found as nuggets, and in small particles in rocks which have never been disturbed from their original positions, but which have been decomposed to a considerable depth, and it then has the same mammillary form, occupying positions which make it evident that it must have been formed *in situ*, and never have undergone any abrasive action. The nugget found in 1828, in Cabarrus County, N.C., which weighed 37 lb., and also the one found in the valley of Taschku Targanka, near Miask, in Siberia, which weighed 96 lb., were both found under such circumstances in a decomposed dioritic rock. In some few

* For a detailed list of the fineness of California placer and quartz gold see Min. Com. Rept., 1870, p. 768.

cases it has been definitely ascertained that the gold has been dissolved and precipitated in the decomposed rocks, for it has penetrated only just so far as the decomposition has allowed it, the yield in gold ceasing entirely at the point where the rock allowed no further filtration; while in other rocks of a more porous nature in the same district the gold has penetrated to a depth not yet ascertained. Such a condition of things is not uncommon in the gold belts of the Southern States.

Admitting that heavier masses of gold did exist in the veins disintegrated by the ancient rivers, gravity alone cannot account for the bottom deposits, which are often 300 ft. and sometimes 500 ft. from the surface, being the richest. It would have required greater agitation of the earth than we have any evidence for believing ever took place to sift the coarse particles through even 50 ft. depth of earth, and there is no indications that these deposits after they were once made were ever disturbed. It is undoubtedly true that in shallow placers, where the bedrock comes near the surface, the surface soil is rich, but it is the invariable rule that in the deep placers the richest deposits are near the bedrock, and at a great distance from the surface.

There is a tradition which is prevalent in all the shallow placer gold mines of the south, and in those of some other districts, to the effect that gold *grows* from the *seed gold* which is not extracted, so that every few years the tails of the old mines are re-worked, generally with a profit; the quantity separated each time, according to the local tradition, being in proportion to the length of time the material has remained undisturbed. As there is no opportunity for the gold in these sands to accumulate by gravity, the people of the region believe that gold grows like a plant.

It would not, however, be rational to deny a theory so easily explained as the formation of placers by the destruction of vein matter without having some other to replace it. If the theory of the destruction of pre-existing veins is not tenable in all cases, we are bound to examine carefully whether there are causes in nature sufficient to account for solution, and what are the agencies that render the gold soluble. I have made a series

of both synthetical and analytical experiments on this subject lasting over many months, which seem to be of considerable importance in the study of the origin both of placer and vein phenomena. In this investigation most of the known salts of gold were prepared; but as the chloride is most easily made, this was made the basis of almost all the solutions. While making the gold chloride for the solutions some sponge-gold was placed in a tube and heated in a current of chlorine gas until the gold chloride formed was entirely sublimed. It deposited at the upper part of the tube directly over the gold, as the tube cooled, and on the gold also, in fine transparent crystals half an inch long. This tube, when cool, was closed while full of chlorine, by replacing the glass tubes by glass rods, and the joints made tight with paraffine. In five months the crystals were melted into a mass, and in a year the whole of the chloride had been transformed into metallic gold with occasional nodules of chloride through it.

In order to ascertain the effect of different organic substances on salts of gold in solution, five portions of 50 cubic centimetres each of a solution containing .50 gramme of gold chloride were treated in different ways. The first was covered with a cubic centimetre of petroleum. In the second a quarter of a gramme of cork was placed; in the third a quarter of a gramme of peat; in the fourth half a gramme of leather; in the fifth half a gramme of leaves. These solutions were put in a dark place, and were left for three months before examination. When the solution containing the petroleum was brought to the light the liquid had lost its colour, and there were suspended in it a number of very fine and long crystals of gold, distributed nearly uniformly from the top to the bottom, and floating almost perpendicularly in the water. They had the appearance of the hexagonal crystals described by Professor Chester.* As soon as the liquor was agitated they fell to the bottom. The solutions containing the cork, leather, and leaves had also been rendered colourless, but the gold had entered into these substances, replacing the organic matter, so that they were pseudomorphed into gold. The solution in which the peat was placed was also colourless, but the gold was preci-

* American Journal of Science, 3rd series, vol. xvi., p. 32.

pitated in the form of very small mammillary masses, recalling perfectly the form of nuggets.

To ascertain the degree of solubility of gold a quantity of pure spongy gold was prepared and placed in a variety of solutions; some of these were left exposed to the air; others were sealed at the ordinary temperature and pressure of the air for periods of from six to eight months; others were exposed to heat and pressure under varying conditions in an air-bath, arranged in such a way that the temperature could be kept constant for a great number of hours at a time. Many of these last tubes burst after the liquid had acquired a tint. Of some of these the contents were entirely lost, of others a sufficient quantity of the liquid was left to test for gold.

Solutions of salt, ammonium sulphate, ammonium chloride, potassium chloride and bromide, in sealed tubes, after eight months gave no reaction. Heated for five hours at temperatures varying from 150 deg. to 200 deg. Cent., none of them, except the potassium bromide, gave any reaction, and that reacted very strongly. In the sealed tubes the solution of salt, in which a few drops of nitric acid had been placed, reacted for gold; the potassium iodide gave no immediate reaction, but when evaporated to dryness left a purple residue, soluble in bromine, which reacted for gold. Heated to a temperature of from 100 deg. to 170 deg. Cent., the potassium iodide tube gave a reaction for gold not much stronger than the solution before heating.

A solution of commercial ammonium nitrate, which contained some ammonium chloride as an impurity, kept in an open tube at the ordinary temperature and pressure for four and a half months, coloured the solution bright yellow, and reacted strongly for gold. Two solutions were made, each containing 5 grammes of ammonium nitrate and ½ gramme of ammonium chloride in 200 cubic centimetres of distilled water. One of the solutions was left in an open room and the other put in a dark place and left for eleven days. At the end of that time both reacted strongly and with equal intensity for gold.

Pure sponge gold was then placed in the following solutions, contained in sealed tubes at the ordinary temperature and pressure for three months. Ammonium sulphide produced no change

and no reaction. With potassium sulphide a black precipitate was formed, and a strong reaction for gold was given by the liquid. Sodium sulphide gave a black precipitate and a strong reaction for gold. Potassium cyanide gave a yellow solution, a brown precipitate and smell of ammonia, and a strong reaction for gold. Magnesium chloride, after nearly three months, gave a gelatinous precipitate, but no gold. Sodium sulphate, after the same length of time, produced no change and no reaction. The copper sulphate produced no change after two and a half months.

Spongy gold was then put into solutions of the following substances, and heated for six and a half hours between 145 deg. and 180 deg. Cent. Ammonium sulphide showed no apparent change, but reacted strongly for gold. The solution of potassium sulphide attacked the glass strongly; it looked greenish; and the liquid reacted for gold; a black precipitate was formed, which was dissolved in bromine, and reacted for gold. The solution of sodium sulphide acted slightly on the sides of the tube, and acquired a greenish tint; a pink film was found on the glass and a slight precipitate was formed. This film reacted slightly, and the solution very strongly for gold; there was not enough of the precipitate to examine. The solution of magnesium chloride attacked the glass strongly, from which scales fell, but no gold was dissolved. The solution of sodium sulphate gave a cloudy flocculent precipitate, but no reaction for gold. Commercial sulphuric acid and solutions of potash, iron, and manganese sulphate, gave white scales, but no reaction for gold. Solutions of sodium sulphate and nitrate gave no change and no reaction. The solution of potassium permanganate produced no reaction. In the solution of potassium cyanide the brown precipitate which was formed in the previous experiment dissolved, reducing the gold in the solution so that no gold was found dissolved.

A mixture of silver nitrate and sulphuric acid produced no change after two months. A mixture of the potassium, iron, and manganese sulphates, and commercial sulphuric acid, produced no change after two months. The potassium permanganate and sulphuric acid gave a black precipitate and coloured the liquid slightly pink, but gave no gold.

In order to test the effect of organic matter in solution, half a gramme of gold chloride was placed in 2 litres of Croton water in two large bottles. One of these was left exposed to the sunlight, and from this all the gold was precipitated in less than a week; the other was placed in a dark room and left there; at the end of eight months a small amount of gold was precipitated. When solid organic matter was placed in the bottle the precipitation was quite rapid, and when the bottle was then brought into the sunlight all the gold was precipitated in about forty-eight hours.

To ascertain the effects of the different soils on weak gold solutions ½ gramme of gold chloride was dissolved in 10 litres of filtered Croton water, and made to pass continuously over the three mixtures given below arranged in glass funnels. The apparatus was so arranged that the liquid would flow drop by drop on the filters.

No. 1 contained 30 grammes of quartz sand; No. 2, 20 grammes of sand and 10 grammes of soil; No. 3, 30 grammes of magnetic iron sand and 10 of quartz sand.

The filters were left exposed in a room where there was considerable dust arising, and where there was also the smoke from passing trains. In two days most of the gold had been precipitated in the filters, and the water had a greenish look; ½ gramme of gold chloride was then dissolved in 10 litres of distilled water, and filtered in the same way over 30 grammes of quartz sand, a mixture of 20 of sand and 10 of soil, and a mixture of 10 of sand and 30 of magnetic sand. These filters were carefully covered so as to prevent any dust settling on them, so that they were protected from all organic matter except such as was contained in them. At the end of two months the clean sand and the mixture of magnetic and clean sand had reduced a small quantity of gold, a little more in the latter than the former, in concretionary shapes, which, owing to the rapidity of the action, were not coherent, but could be crushed with the pressure of the finger. In the mixture with the soil the whole had been reduced, and was distributed through the sand as an impalpable powder, no indication of any concretionary form being observed.

The attempt was then made to dissolve gold in a manner

similar to that which was supposed to take place in nature. For this purpose filters were prepared of 30 grammes each of clean quartz sand; in one of these 1.161 grammes of sponge gold was placed and carefully mixed with the sand; in each of the other two, $\frac{1}{2}$ gramme of very fine gold was mixed. Over the sponge gold 10 litres of distilled water, containing 30 grammes of common salt and 5 grammes of sodium nitrate was made to filter constantly for two months, but no observable change took place. For the second solution 6 litres of Croton water were taken, in which 9 grammes of ammonium nitrate and 1 gramme of ammonium chloride were dissolved. This was made to filter constantly for one month, but no gold was dissolved. For the last experiment 1 gramme of ammonium nitrate and 9 grammes of ammonium chloride were used, but no gold was dissolved.

It was the intention to continue these filtrations for six months at a time, and with all the conditions of natural waters, but the difficulty of making the experiments continuous decided the abandonment of them after a number of the other results had been obtained. The failure to dissolve gold in this short time does not prove that there is no action, as the other experiments show. An amount equal to a little less than that in sea-water might easily escape detection. In these experiments there is lacking the certainty that all the conditions necessary to success will be fulfilled. It was found in one of the experiments, made in the early stages of the investigation, that the dust circulating in the room, invisible except in a strong beam of sunlight, was sufficient to precipitate the gold from a dilute solution. All these researches had to be made in a room to which many persons had access, and it is quite possible that the organic matter precipitated the gold in these last experiments as fast as it was dissolved; for in the experiments for the production of the placers the organic matter did deposit the gold in the sand. It is greatly to be regretted that these experiments could not have been made in the complete absence of dust, and in a room in which the air could have been filtered.

In order to test the effect of organic life in such solutions, a plant was watered with a very weak solution of gold, but as is often the case in such researches, from the great interest of my

assistants, and the intense desire that the experiment should succeed, so that gold might be recovered from its ashes, the plant died of too much watering. In the anxiety to produce the kind of absorption by plants described by Durocher and Malguti, the experiment was made a failure by too much enterprise The examination of the ashes of the plant showed a small amount of gold, but most of the gold precipitated was in the soil around the plant, being thrown down there, by the organic material in the earth. This experiment indicates the origin of the thin plates of gold which are sometimes found in the grass roots of certain placer countries.*

It will be observed that in almost all the cases where gold was dissolved, chlorine and some nitrogenous substances were found together in the presence of alkaline waters. These same conditions are favourable also to the separation and solution of silica. It has been proven by these experiments that the alkaline sulphides act on gold as well as the substances enumerated above, and it is quite easy to imagine the conditions under which the gold, already in solution in excessively small quantities, coming in contact either with solid or liquid organic matter, may precipitate all the metal.

In Grass Valley, California, I have known of gold being thrown down in the filter of a Plattner's vat by the organic matter contained in the very impure water used in those works, for the solution of the gold rendered soluble, by the action of the chlorine. The filter was full of metallic gold, and there were no means of ascertaining how much of it had been lost. I examined several ounces of a brown deposit taken from it which was nearly pure gold. This cause of deposition, and of loss in large operations, has, I believe, been entirely overlooked. It is quite easy to explain the presence of gold in alluvial sands by the action of sunlight alone on the waters containing the gold in solution, and to account for the gold on the bedrock by the solutions coming in contact with organic or mineral matter, such as the lignites, fossil woods, or the pyrites, which are everywhere found in deep placer deposits. The waters, not being able to pass the bedrock, remain there in contact with the

* Trans. Am. Inst. Min. Eng., vol. vi., p. 33.

organic matter until all the gold is precipitated. The same would be true of the decomposing rocks, or of slaty strata coming up to the bottom of the deposit at an angle. The deposition would be rendered much more rapid by any electrical currents that might pass through the strata.

In all of these phenomena time, which in the operations of nature is unlimited, is one of the chief factors. In any laboratory experiment the limit of time must of necessity be short, but there is no such limit in nature. That this solution goes on on a large scale there is every reason to suppose. That it may be connected with vein phenomena, the California nugget shows, since in this case both the formation of the quartz and of the nugget are evidently posterior to that of the blue gravel.

It will be seen from these reactions that many of the conditions favourable to the solution of gold are also favourable for the solution of silica, and that, as Professor Kerr shows,* the rocks may be actually decomposed and the gold deposited, forming in this way shallow deposits which are improperly called veins, in which the gold disappears entirely beyond a few feet.

Nothing is more likely than that the infiltration of water through rocks undergoing decomposition, of which there are enormous quantities in the gold region, should take up the alkalies, and, slowly passing over the gold, should dissolve it. The composition of these alkaline salts would depend on the nature of the rock through which the waters passed, but it is more than likely that they would be mixtures of many of the compounds likely to attack the gold and carry it off in solution, and not alkaline carbonates and sulphides alone, although these would be likely not only to be present, but to be powerful agents in carrying on the work of solution. In some cases the decay of the rocks is so rapid that the phenomena may, as it were, be caught in the act. The agencies producing the decomposition of the rock, penetrating it beyond the limits of local drainage, and, carrying off the soluble parts, leave the débris in a condition easily penetrated by the infiltrating solutions, and ready to receive any deposit which these solutions may from any cause leave behind them. A source of these deposits in the deep

* Trans. Am. Inst. Min. Eng., vol. viii., p. 462.

placers of California is the trap which sometimes covers the old river deposits to a depth of 150 ft. In the deep placers these waters would be capable of holding the gold in solution until they met some decomposing element, such as particles of metallic compounds, native metals or organic matter, which is always present in large quantities in the deep placers. If a nucleus of metal were present the gold would be precipitated on it, and if none were present then the gold would come down as a powder, each grain of which, however small, would serve as a nucleus for future aggregations.

Admitting the solutions to be even more dilute than the sea-water near the coast of England, yet unlimited time and infinite quantity, even of very dilute solutions, would evidently produce these effects, redistributing the gold whenever there was anything in the sands which would precipitate it. Iodine, which is a solvent of gold, is found in many of the plants of the gold region, and in considerable quantities in sea-water. Sonstadt* supposes that gold is kept in solution in the water of the sea by the slow rate at which calcium iodate is decomposed in the sea-water of the temperate zones, but suggests that where the decomposition of the iodate, whose presence is necessary to keep the small amount of gold in solution, is very rapid, as in hot countries, the liberation of the nascent iodine, and consequent rapid solution of the gold, and subsequent precipitation by organic matter, is quite sufficient to account for the great richness of the gold deposits of tropical countries.

It seems, by the experiments already cited, to be clearly proved that gold is not only not insoluble, but that in nature it is constantly being dissolved out of the rocks and placers, the waters of filtration dissolving out of the rocks, in their passage through them, all the materials necessary for the solution of the gold, and carrying it in very dilute solutions until it meets some substance that precipitates it. It seems to be proved that when the action is slow and localised, we have the phenomena of placers with large or small nuggets and irregularly shaped pieces, and when the action is rapid we find the gold in small particles distributed through the sands. We have reason to

* Chemical News, vol. xxvi., p. 161. Am. Chemist, vol. iii. p. 208.

suppose that these phenomena are now taking place in su way as to concentrate the gold by infiltration and precipit in the tailings of mines which cannot be concentrated mechanical means. Some of these phenomena can be accou for by the simple action of sunlight, but others, mostly thos the deep placer deposits, have their cause in the large amou organic material contained in them. The use of a charcoal f to precipitate the gold from relatively concentrated solution one of the recently invented metallurgical processes, is a suggestive idea of the means nature may have used or immense scale on very attenuated solutions.

The same conditions which cause the solution of gold in ce cases cause also the solution of silica. This explains the ph menon of mammillary and apparently water-worn nuggets, that from Placer County, encased in quartz, while both the and the quartz have been formed posterior to the blue gr It also explains the presence of "putty stones," as the soft p of decomposed rock constantly found in placer deposits called. Many of the causes which produce the precipitatio the gold would also produce the reduction of soluble sulphat insoluble sulphides, the gold being retained in the mass. would account for the almost constant presence of gold in py or the occurrence of some of the copper ores of Texas in the f of trees, the ore containing both gold and silver, and also fo constant presence of gold in the iron ores of Brazil, the so-c Jacotinga, and also for the presence of trees transformed iron ore carrying gold in some of our Western States. In n of the deep placers of California the heavy cap of basalt is c sufficient to account for many of the phenomena which c not only beneath but around it.

The fact that gold has not as yet been found in potable w may be simply due to the extreme difficulty attendant o detection in minute quantity. It is more than likely that n of the geological phenomena on a large scale were produce very dilute solutions or very slight forces acting for a very time. How far the electrical currents of the earth may have a factor in these phenomena it is impossible to surmise.] however, more than probable that they were the result, n

one alone, but of all the causes mentioned, and perhaps many others which have as yet escaped our attention. No single agent is so powerful a solvent of gold as chlorine. Very few drainage waters are free from some compound of it, and no soil is without the nitrogenous materials necessary to set the chlorine free, and therefore capable of attacking the gold and rendering it soluble. The experiments show that a trace of it is quite sufficient to dissolve enough gold to colour a solution so that the eye can detect it after a few weeks' exposure.

In the nugget of Placer County it would have been impossible for either the gold or the silica to have got into its position except by solution. The iron of the blue gravel in this case seems to have been the first cause of precipitation, and subsequently the gold itself was an active agent in increasing its own weight, causing a constant precipitation on its own surface of the small quantities of gold contained in the waters passing over it. The general absence of crystals and their rounded edges, where they are found, can be easily accounted for by the fact of the rapid action possible in the placers. The readiness of filtration through the relatively easily permeated gravel causes the gold to precipitate so rapidly that there is no time for any but a mammillary deposit; while in vein deposits the extreme slowness of the deposition allows the gold to assume a crystalline shape. When we consider that two-thirds of all the gold produced in the world comes from alluvial deposits, it seems difficult to account for its presence in the sands in any other way than by solution.

Some years ago experiments were made to ascertain whether small thin pieces of gold which had proportionately greater surface did not contain less silver than the coarse particles, with the following results:*

	1. Coarse Gold.	2. Coarse Gold.	3. Fine Gold.	4. Gold Bullion.
Gold ...	.902	.893	.917	.915
Silver ...	.098	.107	.083	.085

This is interesting in connection with assays made on the Homestake Mines, which show that the proportions of silver

* Trans. Am. Inst. Min. Eng., vol. x., p. 470.

are very much greater in this vein than in any placer s
mens.

	Homestake.	Highland.	Terror.	Deadwood.	De Sm
Gold... ...	.820	.830	.825	.850	.820
Silver ...	.170	.155	.160	.140	.170

On account of its very great interest, a list of such large n
gets as have been described with moderate accuracy is anne
They have been compiled from all the public statement
which I could get access.* There are undoubtedly some w
are apocryphal and have existed either only in the imaginatio
those who described them, or have been so constantly, and
increasing exaggeration, described, either by tradition or
newspaper press, that they have become almost as apocry
as if they had never existed. There are also some which
doubtedly appear upon the record twice, either because t
weights have been inaccurately stated or because their v
has been estimated from weight or weight from value. I
surprising how little we know about the real value of an
these nuggets. This is owing partly to the fact that it c
hardly be known until they were melted, and then the ow
of them having realised all they could, either made no recor
it, or for some reason did not state the real value, and partl
the fact that the value was estimated by weight as pure
when the fineness of the nugget was much below it. It is alm
as difficult to get the true history of these nuggets as it is to
the history of the discovery of gold. It is scarcely possible
this list should contain all of the nuggets that have been
scribed, and it is certain that it does not contain all th
that have been found. There is a tradition and some evide
to substantiate the fact that in the early days and more recen
for fear that the real or supposed value of certain claims sho
become known, thereby endangering the ownership of the cl
and for fear of its being seized, or, as the miners say, "jump
large nuggets which have been found have been cut up be
being sent to be melted, for fear that others might discover
locality that produced them.

* Reports of the Mining Commissioner of the United States, Washing
1867 to 1876. Production of Precious Metals in the United States, 188
1884.

LIST OF SOME OF THE PRINCIPAL NUGGETS FOUND IN THE UNITED STATES, AND ELSEWHERE.

Date.	Locality.	Weight.
	CALIFORNIA.	oz.
1854	Carson Hill, Calaveras County, near Angel's Camp quartz vein, 195 lb., 4 lb. quartz*[1]	2340
1854	California, Calveras County	1920
1850	Holden's Garden, Sonora, Tuolumne County. A mass of quartz and gold*[2]	1818
1860	Monumental quartz, Sierra Buttes, Sierra County. Found by W. A. Farriah*[3]	1596
1871	Rattlesnake River, California	1272
	California, Sierra County	1272
1871	Kanaka Creek, Nevada County, California	1152
1869	California, monumental quartz mine, Sierra County	1146
1848	Wood's Creek, below Sonora, Tuolumne County	900
1871	California, St. Charles Hill, Sierra County. Not authenticated	768
	Willard Claim, west branch of the Feather, Butte County, 54 lb. before melting and 49 lb. after melting	648
1869	Monumental Mine	1146
1855	French Ravine, Sierra County*[4]	532
1873	Bald Mountain	65
1873	Bald Mountain	40
1867	Pilot Hill, El Dorado County. Boulder of gold quartz	426
1851	French Ravine, Sierra County	426
1849	Sullivan's Creek, Tuolumne County, 28 lb.	408
	Knapp's Ranch, east of Columbia, Tuolumne County. A slab-shaped mass, 14 in. by 9 in. by 5 in., which, with other fragments weighed*[5]	396
	California, Sierra County	384
1849	Gold Hill, near Columbia, Tuolumne County	360
1876	Polar Star claim, Dutch Flat, Placer County, from a white quartz boulder*[6]	349
	Spring Gulch, near Columbia, Tuolumne County. Globular form*[7]	303
	Kelsey, El Dorado County, or near it; date not stated*[8]	285
1853	Columbia, Tuolumne County	283
	Minnesota, Sierra County	266
1850	French Ravine, Sierra County	263
	Mokelumne River, Calaveras County; said to weigh from 20 lb. to 25 lb.	240 to 300
1864	California, Georgetown, El Dorado County	226
1865	Michigan Bluffs, Placer County, two miles distant on American River	204
1855	Remington Hill, Nevada County, estimated to weigh	186
1866	Smith's Flat, Sierra County	146
1864	Smith's Flat, Sierra County	140
	De Groots, Terry, and Klamath Rivers, Siskiyou County	131
1867	Remington Hill, Nevada County	128
1865	West branch Feather River, near Magalia, Sierra Co.*[9]	107
1869	Little Grizzly, Sierra County	107
1865	Spaniah Dry Diggings, El Dorado County, Gritclaim, dendritic gold. Found by Fricot	101.40

* Estimated value.

[1] $43,534. [2] $30,000. [3] 17,655. [4] $10,000. [5] $8500. [6] $5760. [7] $5000. [8] $4700 [9] $1760

Date.	Locality.	Wei
	CALIFORNIA—*continued.*	
1862	West branch Feather, near Magalia, Sierra County. Found by Morrison*[1]	
	California, Nevada County	
1869	Hope Claim, Sierra County	
1860	French Ravine, Sierra County	
1854	Camp Seco, Calaveras County, Stone Cabin Gulch. Found by Frank Russworm*[2]	
1861	Smith's Flat, Sierra County	
1850	Oregon Cañon, El Dorado County; supposed to be the Fay nugget*[3]	
1854	Spanish Dry Digging, El Dorado County, near the Grit seam, Texas and Jacobs*[4]	
1849	Illinois Cañon, near Georgetown, El Dorado County*[5]	
1850	Georgia Slide, Hudson's Gulch, El Dorado County*[6]	
1865	Lowell Hill, Nevada County	
1860	Spanish Dry Diggings, El Dorado County, Pennsylvania seam, 2½ lb.	
1857	Garden Valley; El Dorado County; found by Samuel Treeworgee*[7]	
1854 } 1862 }	Live Yankee claim, Forest City, Sierra County; 12 nuggets from 30 oz. to 170 oz.	{ 1
	Oregon claim, Forest City, Sierra County; nuggets from 30 oz. to 100 oz.	{ 1
	Banghart Mine, Shasta County, Mad Mule Cañon; crystalline. Found by Cooper	
	NORTH CAROLINA.	
	Reed Mine	9
1838	Cabarrus County	4
1835	Cabarrus County	1
1824	Cabarrus County	1
1829	Anson	1
	URAL MOUNTAINS.	
1842	Near Miask, Siberia	11
1878	One	9
	One	3
	Several, each	1
	SOUTH AMERICA.	
	Paraguay	6
1730	Peru	7
	AUSTRALIA.	
1852	Meroo Creek, New South Wales, "King of the water-worn nuggets"	–
1850(?)	Sarah Sands	26
1853	Ballarat, Victoria	22
1858	"Welcome," Bakary Hill, Ballarat, Victoria	21
1842(?)	Blanche Barkley	17
	"Blanche Barkley Nugget," Victoria, with 6 oz. quartz, weighed 146 lb.	17

* Estimated values.

[1] $1607 [2] $1760 [3] $1250 [4] $1100 [5] $1000 [6] $1000 [7]

Date.	Locality.	Weight.
	AUSTRALIA—*continued.*	oz.
1853	Ballarat, Victoria	1620
1849	Two, each	1344
	Buwanding, New South Wales, near Orange, mixed with quartz and iron pyrites, gross troy weight 107 lb. 2 oz. 8 dwts.	1286
1851	Meroo Creek, New South Wales, three, each	1272
1870	"Viscount Canterbury," John's Paddock, Victoria	1105
1855	Maryboro', Australia	1060
1855	Fryer's Creek, Australia	1008
1870	"Viscountess Canterbury," Berlin diggings, Victoria	884.10
1855	Old Daisy Hill, Australia	860
1851	Meroo Creek, 53 miles from Bathurst, New South Wales, weighed 70 lb. and gave 60 lb. troy of gold	840
1851	Meroo Creek, New South Wales, two, each	720
1871	"Rum Tow," Catto's Paddock, Berlin diggings, Victoria	718.5
1855	Ballarat, Victoria	576
1852	Bendigo, Victoria	564
1872	"Schlemm," Victoria, including 60 oz. quartz	538
1855	Ballarat, Victoria	480
1874	McGuiggan's Lead, New South Wales	444
1874	McGuiggan's Lead, New South Wales	410
1861	"Platypus," Robinson Crusoe Gully, Bendigo, Victoria	377.6
1851	Meroo Creek, New South Wales, 30 lb. 6 oz.	366
1851	Meroo Creek, New South Wales, 24 No. 1. The Brenan nugget measured 21 in. in circumference	364
1852	Bendigo, Victoria	340
1852	Dascome	336
	Whipstick Flat, Kiandra, New South Wales	336
1874	McGuiggan's Lead, New South Wales	300
1855	Mount Blackwood	288
1871	"The Needful," Victoria	246.16
1858	"Beauty," Kangaroo Gully, Bendigo, Victoria	242
1861	New Chum Hill, Kiandra, New South Wales	200
1872	"Crescent," Berlin diggings, Victoria	176.8
1873	"El Dorado," Victoria	170
1860	Sunoy River, Kiandra, New South Wales	160
1852	Meroo Creek, New South Wales	157
1882	Temora, New South Wales	153.17
1860	Tooloom Diggings, New South Wales; nearly solid gold	140
1874	McGuiggan's Lead, New South Wales	134
1872	"Spoudulix," Eureka Gully, Jordan's, Victoria (29 oz. quartz)	130
1873	"Alma I.," Victoria	125
1880	Temora, New South Wales	99
1860	Kiandra, Snowy River, New South Wales	93.18
1880	Temora, New South Wales	84
1874	McGuiggan's Lead, New South Wales	84
1852	Louisa Creek, New South Wales, a solid lump	82
1875	"Lothair," Clune's, Victoria; at 307 ft. depth	77.6
1880	Temora, New South Wales	76
1880	Temora, New South Wales	72
1852	Meroo Creek, New South Wales	71
1880	Temora, New South Wales	68
1874	Wood's Flat, New South Wales	65
	Gundagai, New South Wales, new diggings	64.7

Date.	Locality.
	AUSTRALIA—*continued.*
1876	Canadian Lead, near Gulong, New South Wales
1880	Temora, New South Wales
1880	Temora, New South Wales
1880	Temora, New South Wales
1875	Buninong, Victoria
1874	Wood's Flat, New South Wales
1857	Laura Creek, New South Wales, gold and crystallised quartz
1872	Broomfield Gully, Creswick, Victoria
1875	Sandhurst, Crusoe Gully, Victoria
1879	Nundle Goldfields, New South Wales
	New Chum Hill, Kiandra, New South Wales
1874	McGuiggan's Lead, New South Wales
1876	McGuiggan's Lead, New South Wales
	Oldham nuggets, Turton's Creek, Victoria, from 2 oz. to 36 oz.
1874	McGuiggan's Lead, New South Wales
	Broad Gully, Braidwood district, New South Wales
1871	"Nugget," Creswick, Victoria
	Ballarat, Gold Reef claim, Victoria
1872	"Red Streak Lead," Victoria
1873	Upper Boggy Creek, Victoria
1852	Bingera, New South Wales
1874	McGuiggan's Lead, New South Wales
1872	Broomfield's Gully, Creswick, Victoria
1879	Nundle Goldfields, New South Wales
	McGuiggan's Lead, New South Wales
1876	Wapping Butcher, near Parkes, New South Wales
1873	"Alma 2," Victoria
1880	Nenigundah, New South Wales
1851	Summer Hill Creek, New South Wales, one of the first found
1874	McGuiggan's Lead, New South Wales

CHAPTER II.

HYDRAULIC MINING.

WHEN the surface placers grew poor, no longer yielding the very large amounts of precious metal near the surface, which could be extracted by simple means, the miner worked deeper and washed poorer claims, which had previously been unnoticed, and which he followed until they led him into unexpected difficulties. When he worked a rich gravel lead, it often carried him under cover of so much unproductive dirt that he had to strip the ground or work the bank either with tunnels or shafts, or abandon it altogether, according to his ability to contend with and master the difficulty. Finding, therefore, that he was working poor ground in combination with rich, he undertook to work the poor grounds, or what were to him then poor grounds, when there was no very rich pay beneath it, because they were near the surface and easily accessible. Afterwards he worked the rich without the poor. Where ground to be worked could be reached easily from the surface it was stripped, and ground-sluicing afforded a means of extracting the gold, but when this ceased to be profitable, shafting and drifting had to be substituted, and as this was always expensive, only the relatively richest deposits could be worked, and the claim was abandoned when the part near the bedrock had been worked out. Very soon distinctions between the different varieties of occurrence of the precious metals commenced to be made. The surface deposits were called shallow placers, and those where the gold was found at any distance beneath it were called deep placers, and these latter according to their situation, were worked by what is known as hydraulic, drift, or shaft mining. They are all, properly speaking, hydraulic methods, and differ only in the way the gravel is first obtained and the scale on which the work was done. In the

hydraulic method, properly speaking, all the gravel is w
In the drift method only the few feet next to the bedrock
is rich is mined, and it is excavated just as if it was an orc
deposit in nearly horizontal beds. The pay dirt is extract
the ordinary methods of mining. It is dumped from the
cars into the sluices and treated exactly as in the hyd
method proper. The drift method is never adopted while
draulicking" is possible. It is only used where for some
that could not be applied, as, for example, when the pay
covered with basalt to such a depth and under such circums
that it could not be treated in any other way.

All of these gold-bearing deposits are found on the w
slope of the Sierra Nevada Mountains. It was here that sl
placer mining commenced, out of which all the other syste
hydraulic mining grew. It was carried out here for a lon
on an enormous scale, yielding the larger part of the gol
was extracted in the United States. At first it was prosecu
a small way only, but it very soon became evident that
working was to be done profitably it could only be do
very large amounts being treated at a time, and conseq
combinations of capital became necessary in order to work
scale required, which necessitated the moving of millions o
of dirt, and disposing of the tails, after the extraction of th
cious metals, in such a way as to be no longer an impediment
miner. And here the difficulty commenced, for when the
was no longer embarrassed by these same tails the agricu
was. The rivers began to be filled up and the low lands
covered with detrital matter to such an extent that the f
finding his personal interests jeopardised, commenced a con
which he was successful, not only to the detriment of the
but to the very great loss of the production of the precious
in the United States. At the time they were enjoined
Courts the hydraulic mines were producing more than one
of the entire gold production of the state of California.

The placer mines of the United States are at an average
tion of over 3400 ft. above sea level, and those of Californi
are at an elevation of about 2600 ft. The secondary nat
gravel deposits in itself implies a lower average altitude tha

of quartz veins,* from which they are derived, though exceptions are met with in the case of veins unrelated to the drift matter, or whose croppings have been largely cut down by erosion, possibly also where the conditions have been reversed by earth movements The large number of shallow alluvial placers on the foot-hills and on the edges of the Sacramento and San Joaquin valleys, not on the lines of ancient river channels, but re-deposited from the latter, are at less altitudes than the average stated, while the beach sands, which are not included in this computation, form the lowest group. The great drifts of the ancient rivers are in places from 1500 ft. to 2000 ft. above the present drainage system. The order of succession of the deposits in point of time as well as of altitude, is, first, the original quartz veins; second, the drift accumulated in the ancient river channels; third, the shallow alluvial placers, a re-deposit or second concentration from the earlier drifts. Above the zone of the ancient rivers, in places, deposits of wash from the quartz croppings of recent origin are found; and, on the other hand, below the old river beds occasional accumulations of gravel occur, which are not re-deposits from the latter, but are formed by the disintegration of veins which reach the surface at a less elevation than that of the ancient channels. Placers are sometimes by erosion left at the summits of hills or mountains, but this is the exception, not the rule, though some very important deposits occur on the crests of the foot-hills. Alma township, in Park County, Colorado, contains the highest placer reported. Its elevation is stated at 10,000 ft. Tables on pages 82 and 83 show the results of an examination for altitude of 174 working placer mines.

These deposits occur in various ways: first, as well-defined river channels of ancient river beds under the capping of lava which in past ages has filled the river beds completely; second, as isolated hills more or less high, entirely detached from others, but evidently forming a part of a system which, not being protected by lava, has been washed out from a more or less level country, until they now stand as detached hills; third, in basins which were large enough to receive and hold the gravel which was washed out of the ancient river beds which were broken up

* Report of U.S. Census, 1880, vol. xiii., p. 178.

ALTITUDE OF PLACER MINES.*

State or Territory.	Total Number of Mines reported.	Under 500 ft.	Over 500 ft. and under 1000 ft.	Over 1000 ft. and under 1500 ft.	Over 1500 ft. and under 2000 ft.	Over 2000 ft. and under 2500 ft.	Over 2500 ft. and under 3000 ft.	Over 3000 ft. and under 3500 ft.	Over 3500 ft. and under 4000 ft.	Over 4000 ft. and under 4500 ft.	Over 4500 ft. and under 5000 ft.	Over 5000 ft. and under 5500 ft.	Over 5500 ft. and under 6000 ft.	Over 6000 ft. and under 6500 ft.	Over 6500 ft. and under 7000 ft.	Over 7000 ft. and under 7500 ft.	Over 7500 ft. and under 8000 ft.	Over 8000 ft. and under 8500 ft.	Over 8500 ft. and under 9000 ft.	Over 9000 ft. and under 9500 ft.	Over 9500 ft. and under 10,000 ft.	10,000 ft.
Arizona	1	...	...	...	...	...	...	...	...	1	...	...	...	...	...	...	...	...	...	...	...	...
California	49	2	1	5	5	5	19	4	5	1	1	1	...	...	...	...	2	1	1	1	3	2
Colorado	10	...	...	...	...	...	...	...	...	...	...	...	2	1	...	...	...	...	...	...	...	...
Dakota	11	...	...	...	3	2	...	1	...	4	2	1	...	1	...	...	...	...	...	...	...	...
Georgia	7	...	1	4	...	...	...	...	...	...	2	3	1	1	...	...	...	...	...	...	...	...
Idaho	14	...	...	...	...	...	...	...	...	2	1	3	4	3	1	...	...	...	...	...	...	...
Montana	2	1	1	...	...	...	1	6	7	6	7	1	...	1	1	1	...	...	...	...	...	...
North Carolina	67	...	4	8	17	7	1	6	7	6	7	1	...	1	1	...	...	...	...	...	...	...
Oregon	1	...	1	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...
South Carolina	1	...	...	...	...	...	...	...	...	...	...	...	1	...	...	...	...	...	...	...	...	...
Utah	1	...	...	...	...	1	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...
Washington	1	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...
Total	174	3	8	17	25	15	20	11	12	14	13	9	7	7	1	2	2	1	1	1	3	2

RANGE AND MEAN OF ALTITUDES OF PLACER MINES.*

State or Territory.	Highest.	Lowest.	Intermediate.	Mean.
	ft.	ft.	ft.	ft.
Arizona	4,000	4000	4000	4000
California	*5,433	†185	2809	2612
Colorado	10,000	7500	8750	9075
Dakota	‡6,000	‡4000	5000	4795
Georgia	3,000	881	1940	1694
Idaho	§5,700	4800	5250	5354
Montana	‡7,000	4200	5600	5484
North Carolina	780	440	610	610
Oregon	7,000	†560	3780	2780
South Carolina	650	650	650	650
Utah	6,000	6000	6000	6000
Washington	2,000	2000	2000	2000

* Small placers in the Bodie district not included.

† Does not include black sand littoral deposits.

‡ Estimated.

§ Small placers in Alturas and Lemhi Counties 9000 ft.

by erosion at the end of the glacial period; and, fourth, in low rolling hills near the base of the sierras and beyond the reach of the lava flow. Some of the richest deposits have been found on the flat lands on the east side of Table Mountain, and were evidently bars of the ancient rivers. The deposits occur generally in basins or channels which correspond to ancient lake or river beds, whose courses generally run nearly at right angles to the present rivers. They follow irregular lines, their courses being just as sinuous and crooked as those of the rivers of to-day. Beside the gold there is often found in the gravels, platinum, iridosmine, diamond, quartz, garnet, epidote, ruby, magnetite, limonite, chromite, rutile, and occasionally cassiterite. The width of these ancient river channels is very variable. San Juan Ridge, at North Bloomfield, is about 200 ft. wide on the bottom, and 800 ft. on the surface, and is valued at $850 per foot, or $4,500,000 per mile. This is a narrow channel, they not infrequently are over 1000 ft. wide.

The Table* on next page gives a very fair idea of the extent of these deposits.

These deposits are now so completely covered in many places

* Report of U.S. Census, 1880, vol. xiii., p. 181.

SIZE OF GRAVEL DEPOSITS.

——	Length in Miles.	Width in Feet.	Greatest Depth in Feet.
Spring Valley Mine, Cherokee	9	5 to 10,000	400
Bonanza Mine, Mokelumne Hill	10	500	
Wingate and Bunker Hill, Del Norte County	150	½ to ¾ mile	
Gold Run, Placer County	...	200 to 1000	450
Polar Star, ,, ,,	...	300	250
Paragon, ,, ,,	...	800	600
Orion, ,, ,,	...	2000	180
Plumas Mining Company	...	2500	120
La Grange	...	2500	300
San Juan to Laporte Lead	27	600	...
Boise Basin, Idaho	14	15,000	250

by recent accumulations, or cut across by modern v erosion, that without exploration one would hardly recog there had ever been any water channels in that directio maps of the gold workings show the courses of these ri distinctly.

The depth of the deep placers is very variable. On t Hill Divide, about Gold Run, Dutch Flat, and You Bet, Bloomfield and elsewhere, on the San Juan Ridge, abou ville and at Cherokee, Butte County, the heights of t vary from 100 ft. to 300 ft. in depth. At some points i County they are deeper. On Buckeye Mountain the supposed to be fully 500 ft. deep. When such great this are reached, the most of the gravel is apt to be of l

When such deposits are accessible for the whole he are washed down from the surface to the bedrock, an treated in sluices. This process is known as hydraul proper. In some cases after the gold-bearing mat been deposited it was covered by eruptions of volcan such as basalt, lava, or ashes, to a depth varying fro 500 ft. The lava flow has in many cases extended areas, and is sometimes of uniform thickness for long Such deposits can only be worked by drifts or shafts, the process of drift or shaft mining. Fig. 10 shows a Two Mile Bar where ancient diorites D and trachy been covered by the auriferous deposit B, and this aga by basalt, A.

Fig. 11 shows a section of the Main Boy Tunnel, cited in the

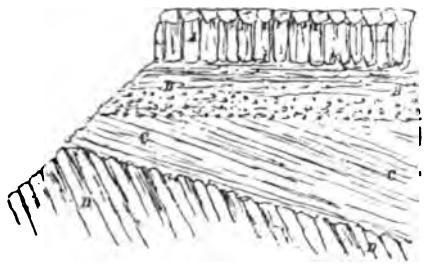


FIG. 10. SECTION AT TWO MILE BAR.

Geological Survey of California, where the whole deposit is covered by a bed of basalt, A, 140 ft. in thickness.

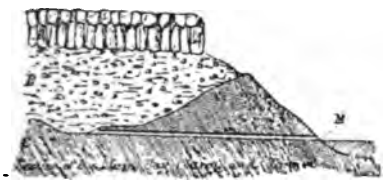


FIG. 11. SECTION OF THE MAIN BOY TUNNEL.

Underneath the basalt is a fine sandstone, B, which readily decomposes when exposed to the air, and below that is the slate, M, inclined at a very steep angle. The "channel" shown in the depression in the centre is 100 ft. wide and from 4 ft. to 5 ft. in thickness. It is reached by a tunnel 1000 ft. long, which it took four years and three months to complete. In some cases, as at the North Bloomfield mines, a part of the deposit only is lava capped, and this is worked by drift mining. The rest is not covered, and is suitable for hydraulic mining.

These ancient rivers had a very irregular grade which varied from 20 ft. to 200 ft. to the mile. They filled their beds little by little until there was sometimes 300 ft. to 400 ft. of detrital

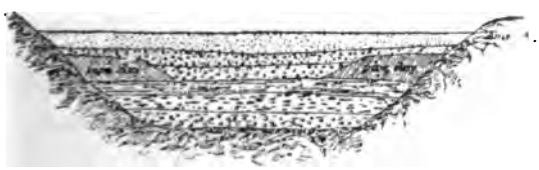


FIG. 12. SECTION OF AN ANCIENT RIVER BED.

matter deposited on the bedrock. Fig. 12 shows such a river bed which had at least three different channels at different

periods of its history. These periods are shown distinctly
figure, the first one showing a regular deposit, the ne
showing a contraction from the deposition of pipe-clay up
borders which narrowed up the channel, the third sh
its full width between the banks. It is conclusively p
by a large number of circumstances, that these river
our modern ones, frequently became blocked, and that
that time there was a considerable deposit of such fine m
as forms clay, which, when the obstruction was broken,
be washed out by the rush of the waters in the centre
the more quiet water near the shore would leave i
paratively unaffected. These were succeeded again by
rivers with less turbulent waters and less obstruction, and
bed has been filled from the bedrock to what is now th
distance sometimes of several hundred feet. The courses
ancient rivers are usually nearly north and south. The

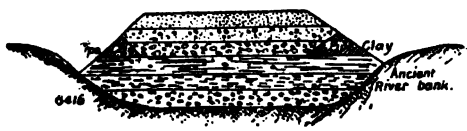


FIG. 13. THE BED OF AN ANCIENT RIVER WITH ITS BANKS ERO

rivers have cut them nearly at right angles, and by thei
have excavated gorges which are sometimes from 200 ft. t
deep, leaving the bottom of the old rivers sometimes 1C
1500 ft. above the beds of those which are now flowir
mile west of Dutch Flat, to the south of Indiana l
North Fork of the American River has crossed the ancient
and completely obliterated it for four or five miles, and
out a cañon 1500 ft. deep to the north of Thompso
Bear River has also crossed it, destroying it entirely
miles and cutting out a cañon 1000 ft. deep.* The
such erosion as this upon the country in which the
channels run, such as is shown in Fig. 12, is distinctl
in Fig. 13, where what was formerly the bottom of a
now become the top of a high hill with deep valleys
side. It is this elevation of the former plateau, owin
enormous erosion which took place undoubtedly at the e

* Mining Commissioner's Report for 1876, p. 61.

glacial period, that makes it now possible to work these old river beds. Not only their courses but their enormous size can be distinctly seen upon the map. The large streams of to-day are but as rivulets and brooks in comparison with those which formed the deposits of ancient times. The channels vary from 200 ft. to 400 ft. wide on the bottom and are sometimes as high as 2000 ft. wide on the surface. They are filled with gravel to depths from 23 ft. to 500 ft. Generally on the bottom there are large boulders and sand firmly cemented. This bottom is generally known as the blue lead from its colour.

The bedrock of these ancient rivers differs with the geological structure of the country. In California, Oregon, Minnesota, Dakota, and Colorado, it is usually a slate or schistose rock. Sandstone and a more or less easily decomposed granite sometimes also occur there. In the Southern States it is often a decomposed mica schist, which not unfrequently allows the gold from sluices to penetrate to a considerable depth and form pseudo veins as well as surface deposits. When the bedrock is regular it greatly facilitates the work. When it is rough or uneven it complicates the collection of the gold and may even throw the workings above or below the grade of the tunnel. Indurated clay sometimes occurs on the bedrock and on the gravel which is so hard that it has to be blasted.

It is estimated that the auriferous deposits have an aggregate length of about 350 miles.* The area is about 200 square miles.

The following sections give a very fair idea of the gravel formations which contain the gold :†

Section across Table Mountain, Tuolumne County.

1. Top. Basalt, overlying andesitic cement 60 ft., forming a bluff 60 ft. high.
2. Bluish soft sandstone, 70 ft.
3. Whitish clays, 10 ft.
4. Auriferous gravels, 2 ft.
5. Bedrock slate.

Section through Shaft 1, Malakoff; North Bloomfield Gravel Mining Company.

Depth of 8 ft. to 10 ft. soil.
To depth of 100 ft. gravel.

* Prod. Gold and Silver in the U.S., 1883, p. 712.
† *Ibid.*, 1881, p. 621.

At 100 ft., streaks of clay mixed with the gravel for a few feet.
,, 120 ft., a foot or two of sand.
,, 130 ft., much quartz; pipeclay disappeared.
,, 140 ft., gravel became more firmly cemented.
,, 175 ft., all the gravel well cemented.
,, 195 ft., thin strata of sand for a few feet.
Then to 205 ft., more cemented conglomerate.

Section at La Grange Mine, Stanislaus County. Top.

1. Red soil coloured by iron oxide.
2. Coarse red gravel, containing granite pebbles, &c.
3. Red cement, hard pan, gravel cemented by iron oxide.
4. White silicious clay.
5. Red cement, as No. 3.
6. Sand with pebbles.
7. Loose yellow sand.
8. Dark-coloured gravel containing débris of granite, slate, serpentine, &c., with some quartz.

The gold deposits, whatever their nature may be, are gravels. They are made up of every kind of [material could be transported by water. They differ both in the s compactness of their component parts. A mechanical ana the gravel in the Nevada Mine showed that it contained 3 cent. of very large pebbles, 29.6 per cent. of smaller s 30.4 per cent. of materials that would pass an inch mesh, o 70 per cent. of coarse gravel. This amount of coarse r would be very largely increased if there were any way o lating the quantity of large boulders always contained in dirt.* A sample taken from the Polar Star Mine at Dut gave 40 per cent. of coarse material, the rest being for t part large boulders. These figures are interesting as s how properly constructed grizzlies and gratings might f the workings of hydraulic claims by separating the mos large material which is sterile, and treating only tha yields gold. The gravels owe their gold almost entirel wearing away of the quartz veins, the detritus of whic the deposit. Occasional evidences, such as have been cite previous chapter, occur, which show that some at least gold has been deposited from solution.

The principal gravel mines of California† are situate

* Prod. Gold and Silver in the U.S., 1882, p. 730.
† *Ibid.*, 1883, p. 713.

counties of Yuba, Sierra, Placer, and Nevada, in the region drained by the Feather, Yuba, and Bear rivers. It has been estimated from careful canvass of this industry that there are on the drainage system of these rivers 300 mines or tracts of mining lands which have been worked from 1880 to 1883, though not perhaps simultaneously, by the method distinctively known as "hydraulicking," besides numerous "drifts," "rivers," and placer mines, which are dependent upon water washing. The magnitude of these mines, whether hydraulic or drift, depends on the width and length of the channel.

The San Juan Ridge, at North Bloomfield, has a channel about 200 ft. wide on the bottom and 800 ft. wide at the surface. This channel, on which the Bloomfield and Milton Companies are operating, has been opened in so many places that its position has been accurately determined and its contents approximately ascertained. The company own about 19,000 linear feet of the channel, which has been continuously worked since 1853—the top dirt having been removed in early times. The value of that channel has been demonstrated to be about $850 per linear foot, say $4,500,000 per mile. To reach the deep or bottom gravel a tunnel 11,874 ft. in length was driven, which was not completed until 1874. Since that time the company has worked 3500 ft. of the main channel, leaving 15,500 ft. to be worked, which it is estimated will produce about $13,000,000. The company's outlet tunnel also commands about one and a half miles of channel, which it may be estimated will produce $8,000,000, or, say, a total of $21,000,000, based on estimates of past production covering a period of about twenty years. Within the limits of the San Juan Ridge alone, in the county of Nevada, it is known that there remains to be abstracted about $90,000,000. In other portions of the county the position of the gravel channels is not at present so well known. It is, however, known that they exist for many miles in length, and the gross yield will vary between $4,000,000 and $6,000,000 per mile.

The claims of the Milton Company have a length of about 17,500 ft., which it is estimated will yield at the rate of $5,500,000 per mile. The tunnels of this company command about two miles more of channel. The contents of this area of

unworked ground will not fall short of $20,000,000. The perty of the Milton Company, including mines, tunnels, voirs, dams, and ditches, cost about $2,000,000. Its tu aggregating more than three miles in length, cost from $6(to $700,000, and its ditches, about eighty-four miles in l cost $462,698, and its reservoirs $214,000.

The North Bloomfield property, including mines, tu ditches, and reservoirs, has cost the present company $2,500,000. Its tunnels cost about $575,000; its rese $246,707, and its ditches $466,768.

The following data respecting these two representative panies are compiled from their books:

*North Bloomfield Gravel Mining Company.**

Expended on mines	$1,595.
Borrowed money	500,
Interest on assessments pending development	800,
	$2,895.

Milton Mining and Water Company.

Expenses from assessments	$1,263
Borrowed on mortgage	300
Interest on assessments pending developments	541
	$2,104
Total cost of both companies	$4,999
Dividends paid on North Bloomfield	$77(
,, ,, Milton	687
	$1,46:

The total payment, up to the year 1883, from net earn dividends and bonds, principal and interest, $2,751,854.

It is estimated on the basis of past production that $25 yet remain in the unworked ground belonging to the panies.

The cost of maintaining the ditches or canals of the panies is an important item of expense. It embraces t of supervision, repair, watchmen, &c. It costs the Milton and the Bloomfield $30,000 per year to keep their ditcl effective condition. The cost of maintenance in the N

* Prod. Gold and Silver in the U.S., 1883, p. 713.

TABLE SHOWING OPERATIONS OF THE NORTH BLOOMFIELD AND MILTON COMPANIES FROM 1877 TO 1883 INCLUSIVE.

Year.	Gold Product.	Water Sales.	Net Profit.	Dividends.
1877	$670,774.57	$6,753.67	$364,045.48	$45,000
1878	849,036.16	9,694.53	501,181.05	484,498
1879	794,517.90	9,090.84	462,821.42	466,390
1880	665.710.60	12,395.26	264,075.49	105,396
1881	637,682.06	24.233.03	325,226.62	
1882	802.190.40	5.134.56	416,208.62	223,424
1883	738,985.97	7,031.64	400,089.77	217,896

NOTE.—The main reservoir of the Milton Company broke June 18, 1883, and the company was deprived of the benefits of half a year's run, thereby materially decreasing the profits. The water stored in the reservoir was ample to continue operations for the balance of the season.

6.10 cents per inch; in the Bloomfield 2.27 cents per inch. These ditches cost originally $7000 to $10,000 per mile. The various reservoirs supplying these mines during the dry season have an impounding capacity of about 5,000,000,000 cubic feet of water. They are situated at elevations of from 5000 ft. to 7000 ft. above sea level. These two mines employ four hundred white miners, at wages varying from $2.50 to $3 per day, and disburse yearly in the immediate neighbourhood of the mines over $300,000.

The total storage capacity of the mining reservoirs of the State is approximately as follows:*

	Cubic Feet.
North Bloomfield Gold Mining Company's reservoirs hold	1,050,000,000
Milton Mining and Water Company, now broken ...	650,000,000
Eureka Lake and Yuba Canal Consolidated Companies	1,130,000,000
South Yuba Water Company	1,800,000,000
Omega and Blue Tent Mining Company	300,000,000
Spring Valley Gold Mining Company	300,000,000
California Water Company	600,000,000
El Dorado Company	1,070,000,000
Small reservoirs on Feather, Yuba, and Bear rivers	700,000,000
Estimates of reservoirs in Amador and Calaveras Counties...	400,000,000
Total	8,000,000,000

The Smartsville gravel deposits are among the most extensive in California. The banks are from 180 ft. to 200 ft. high. The first 100 ft. from the bottom is very hard blue cement,

* Prod. Gold and Silver in the U.S., 1883, p. 714.

mixed with pebbles and large boulders; above this, white pact cement of less coarse material. This bank resiste force of 1400 in. of water under 220 ft. of pressure. Bla was resorted to, and as many as 2000 kegs of black pc have been used at a single blast. This ground has been o at enormous expense by means of long tunnels 8 ft. or 9 ft. and 8 ft. wide, driven through a rimrock of metamorphic at an average cost of $50 per linear foot. The aggregate cos been at least $1,000,000, exclusive of the cost of pipes, flume other incidents connected with a perfect "rig," as it is call the miners. The outfit of some of these claims cost $10,000 to 1882 the Excelsior had run 17,000 ft. of tunnel, and othe panies 4300 ft. The original cost of the ditches bringing on this ground was $11,000,000, exclusive of the cost of maintenance. This channel carried gold from grass-root t rock. Two miles of the channel yielded $14,000,000, claims rim $2,000,000, and there yet remains about one mile of c to be worked. The claims of this district were among th profitable in the State until 1882, when they were temp enjoined, thereby materially diminishing the gold product o

One of the most extensive and productive mining reg California is that of State Creek Basin, situated on the of State Creek, a tributary of the North Yuba. This territory, which embraces a drainage area of 120 squar is situated on the borders of Sierra and Plumas C There are here exposed and partially developed two c aggregating fifteen miles in length. These channels width of from 500 ft. to 600 ft., and a depth of from 3 60 ft. The grade of the old rivers is about 300 ft. to The product of this district has been more than $100,(principally from placer and drift mining. The greater of this sum was extracted by the primitive methods prior to 1862. Hydraulic mining on a large scale is by the water supply, which does not exceed 700(10,000 in. per twenty-four hours for four months in t The drift mines, however, maintain a supply for eight m the year. The auriferous contents of the explored groun

* Prod. Gold and Silver in the U.S., 1883, p. 713.

rims of this basin may be estimated as double the past products. The contents of the unexplored ground is simply a matter of conjecture. Not more than one-fifth of the explored ground has been worked out.

Hydraulic mining is also largely prosecuted, and with great profit, in the counties of Butte and Plumas. In the former is situated the Spring Valley Company, one of the largest and best fitted-out mines in the State. The product for the year 1883 was about $240,000. The results of many years show that the bottom ground pays about $8000 to 100 ft. square, or nearly $35,000 per acre. The company mine banks 300 ft. high, throwing down from 5000 to 8000 cubic yards per day, using 2200 in. of water. A miners' inch at this claim is estimated to be 17,000 gallons in twenty-four hours, and its effect to remove 3 to 4 cubic yards per inch. On the bottom ground it will not exceed 2 cubic yards. The ground pays from the top. This company has expended nearly $400,000 in the purchase of low lands in the Sacramento Valley, which they are reclaiming by filling the depressions with the débris from their mines. This land, once reclaimed, will be a source of revenue to the company.

The mode of occurrence of gold in deep placers is generally quite different from that of the shallow, which are usually richest on top. In hydraulic mines, on the contrary, it is near the bedrock. Where the gravels are capped with lava this concentration of the pay on the bedrock is an advantage, because the drifts are run into it and the upper gravels left, but where there is no such cap it is more expeditious and cheaper to wash down the whole bank. It may be said in general that the yield from the whole surface of a deep placer claim will generally be from 5 to 25 cents per cubic yard. Richer and poorer spots will be found, but they are usually isolated cases. When the general yield of working a very large area has been more than 25 cents or less than 10 cents it may be said to be exceptional.

Where the gold comes to the surface or where a thin upper stratum consists of loam, what is called "grass-root gold" is sometimes found. It occurs much more frequently in shallow than in deep placers, and its occurrence has been recorded in different

countries of both North and South America. It is suppose the finding of such gold as this was the immediate cause discovery of gold on this continent. It is not always s generalise from its occurrence, as it does not always follo because it is found in one place the whole valley in wh occurs will be filled with it. Generally gold found at th face consists of very thin scales which are not easily sav in some few cases small nuggets have been discovered in th of the grass. There seems to be no doubt that in some cas grass-root gold has been precipitated by organic matte solution, but in most cases, it has probably been brought place where it is found, by the purely mechanical inf which affect the distribution of the metal in gravel deposit

In working these gravels if the capital engaged is large e 3 cents to the cubic yard will pay a profit, and 2 cents w expenses. The gold is contained in a detrital material, w called gravel whatever its size may be. A distinction i between the pay on the top, which is generally poor, a bottom gravel, which is usually rich. The constituents gravel are made up of fragments of all the rocks whic ground together at the time the deposit was made top gravel is often a more or less coarse gravel, which and generally of a light colour, but it is sometimes yello reddish-brown from the oxidation of the iron containe Large pieces of lignite are often found in it, which, h fall to pieces when exposed to the air. This top gravel anywhere from 50 ft. to 200 ft. or 300 ft. deep. The lo tions have usually a blue or greenish-blue tint, owing complete oxidation of the iron. It is a conglomerate, mo firmly cemented together. It may be only a few feet, than 100 ft. in depth. The value of the top gravel is mu uniform than that of the bottom, but it is poorer. The b blue gravel, as it is called, is much richer, but there is l Its yield is variable, as it naturally would be, since it fo bed of an ancient river, and while the gold in these streams was deposited on a much larger scale, it wa capriciously left on the natural riffles of the ancient rive now on the more modern ones.

These channels of deposition were all comprised within certain circumscribed limits which were in former ages very distinct, but which have now in many cases been worn away to such an extent that all that is left of the bank of the former stream or lake is sometimes 100 ft. or 150 ft. below the level of the bottom of the ancient deposit, so that what was previously lower than the surrounding country now makes high tablelands with deep ravines on each side. Cases are known where there is a distance of over twenty miles between two such tablelands, which at the time of their formation were one continuous deposit.

The ages of the accumulations are generally tertiary or later, and the deposits may be said to be characterised by the very considerable quantity of fossil wood which has been more or less transformed into lignite, and which, on exposure to the air, generally falls to pieces. All of these ancient river deposits, as well as those of smaller extent and of much more recent formation, are called placers, and are distinguished by the miners as "shallow" and "deep" according to their thickness. The shallow placers vary from 1 in. or 2 in. to 5 ft. or 10 ft. in depth. These are generally worked by hand or with small sluices. Deep placers vary from one to several hundred feet in depth. They cover large areas of ground and are generally inclosed in a basin, the outer edge of which, being of rock, is sometimes called the "rimrock," and sometimes the "channel wall."

In the early days of mining in California the existence of deep placers was not suspected, and consequently only the shallow placers were worked, and when they began to give out, quartz or vein mining seemed to be the only source of gold supply. But little by little it was discovered that gold was not confined to *rich* pay dirt alone, but continued in depth in variable quantities, so that when the top dirt of the rich surface deposits had been worked off by hand there still remained below material which, though it gave "pay" all the way to the bedrock, was too poor to be worked by hand labour alone. Water was then brought from short distances to loosen up the pay dirt, and thus in the spring of 1852, on a claim called Yankee Jim, in Placer County, California, hydraulic mining originated; but the name of the inventor has been entirely lost. Finding that he was not sufficiently remunerated by the

ordinary method of extraction, he constructed a small ditch side of a hill from which he built a flume to the place wh was working. The point where this terminated was 40 ft the ground. The water was here discharged into a common from the bottom of which a hose made of cowhide, 6 in. meter, and ending in a tin tube 4 ft. long, tapering to a 1-in. was attached. The miners in the vicinity were not s imitate and improve upon this process. Indeed, so rapid progress that it was speedily overdone, and in 1857 already fallen into disfavour. It then provided no me working anything but the loose dirt. It was not then se "bank blasting" could be done. For ten years, while th metallurgical industries were being introduced into the ac districts, it almost went out of use. Commencing abou attention was again turned to it, and then little by litt this exceedingly simple beginning the present method of h mining grew. The cowhide was very soon superseded by and finally by iron pipe, though canvas is still used extent in those districts to which sheet iron cannot e transported.

It is estimated that since the introduction of the h methods $100,000,000 have been invested in this me working the auriferous gravel mining in California, an will take fifty years to work out the ground already ope

It is said* this process was known to the Romans, whc ing to Pliny, pursued it on a large scale in ancient S has also been stated that there are evidences in that that it was once inhabited by a large and thrifty pc dependent upon this kind of mining. The remains of dams, and artificial reservoirs are said to have bee showing the extent of the mining prosecuted by thes No one would be more likely to recognise its originality astounded at its immense results than these very Rom would distinctly repudiate, if they could, the idea of ta credit to themselves for it. Since its reinvention in C it has been adopted and carried out on a large scale in Russia, Brazil, and South Africa, to which States Calif

* Prod. Gold and Silver in the U.S., 1883, p. 711.

sent the necessary machinery. It is just as much of an original process in California as if it had never been used anywhere else.

The general operations of a hydraulic mine include water supply and mining proper. This divides the operations of the company into several distinct phases. Under water supply would be:

1. The survey of the ground.
2. The survey of the watersheds.
3. The construction of reservoirs.
4. The construction of the ditch.
5. The construction of flumes.
6. The construction of pipes.
7. The use of the nozzle.

The mining proper consists of:

1. Construction of the tunnel and its connections.
2. Construction of the sluice, which contains:
   A. The undercurrent.
   B. The drop-box.
   C. The grizzly.
3. Bank blasting, and cutting down the banks.
4. The clean-up.
5. The treatment of the gold.
6. The treatment of the tails.

I. The Survey of the Ground.

The deep placer deposits which contain the precious metals vary very greatly in depth, sometimes being within 100 ft., and at others 600 ft. to 700 ft. from the surface. They vary also very much in width, sometimes being only a few hundred and sometimes thousands of feet wide. At Columbia Hill it is half a mile wide. They also vary considerably in quantity of pay, being usually richer towards the bedrock. It is generally true that what are called the top gravels alone, of deep placers, will not pay for any very large expenditure. They not only contain less gold, but it is in a much finer condition, and not easily collected. Thus, at Smartsville the top gravel yielded only $0.02 per cubic yard, and could not be washed alone, while the "bottom" yielded $0.55, and the two together gave a very large profit. The upper gravels of the Gold Run district up to 1874 had averaged, down to the depth of 200 ft., $0.06. The average of the Yuba River district was only a little higher. The nearer the bedrock, as a general

thing, the richer the yield in gold. The best pay is usually from within 10 ft. to 20 ft. of the bedrock, but sometimes as close as 8 ft. Below its level no gold is generally though it very often happens that wherever the deposit is over strata which are inclined, the gold will penetrate i or less, in which case it may pay to enter the bedrock 2 ft. This mode of occurrence is quite different from that shallow placers, which are richest on top. When the gra capped with lava this concentration of the pay near the l is an advantage, because the drifts are run in on it a upper gravels left, but where there is no such cap it expeditious and cheaper to wash down the whole of th It may be said in general that the yield over the whol of a claim will generally be from $0.05 to $0.25 per cub Richer and poorer spots will be found, but they are isolated cases. When the general yield of working a ve area has been more than 25 cents or less than 10 cents, be considered exceptional.

It is, therefore, essential in the commencement of h mining that the ground should be tested from the top g the bedrock. As it is not at all uncommon to find th the top gravel will not yield more than from 1 to 2 c cubic yard, the bottom will give 50 cents to $1 or mor the whole deposit must generally be worked, sheet-iron now used to bring the water, and has completely rep cowhide pipe with the exceedingly simple apparatus of days. Canvas pipe is only used on very small claims or v owners cannot command sufficient capital to set up th "outfit." The inch discharge pipe has been replaced b nozzle discharging under a pressure of 200 ft. to 300 alterations of the surface have so disguised the country idea of its former topography can be gained without a ploration. For this reason preliminary topographica must always be made before starting a claim. As t opening such a mine may be counted by tens of tho dollars, it will be necessary to ascertain first that the rich enough to warrant mining, next that a dump enough to contain all the tailings can be found, that a

grade can be established to carry off all the tails, and that abundance of cheap water can be had as a means of transporting them; and, lastly, that the claim is extensive enough to last a time sufficiently long to insure the repayment of all the capital invested. Preliminary shafts must be sunk to ascertain the depth of the deposit and the nature of the bedrock, which must be found and explored, and the value of the top, middle, and bottom gravel must be learned. It generally happens that it becomes absolutely necessary to undertake expensive constructions, in the shape of tunnels and sluices which will go through the hills, and allow of a discharge at a lower level than the bedrock of the claim, in order to get room both to treat the pay and to carry off the tailings. These tunnels are generally from 6 ft. to 8 ft. in width and 7 ft. in height, and cost from $15 to $75 per foot. So important to the success of the undertaking are they that some of them have taken from four to five years to build. Upon the original location and the inclination of these tunnels and sluices the success of the enterprise depends, other things being equal, more than upon the yield of gold. Where failures have been made it is generally because these surveys had not been undertaken in sufficient detail and with sufficient exactness to secure the proper location of the preliminary works.

The cost of opening a claim alone, not including any of the expenses for water, will often reach as high as $60,000, $70,000, $80,000, or even $100,000, and there are instances recorded where the whole of the money so expended has been totally lost by want of care in the location of the preliminary work. The expenses of the North Bloomfield Company for the year 1871-2, while its preliminary operations were being prosecuted, and before its bedrock tunnel had been driven, were $45,000, not including interest on the capital. It took out $54,000 in gold, leaving a profit of $9000. If it had at this time sold its water at $0.16 the inch for twenty-four hours it would have made a profit of $40,000, but the opening of its own mines would have been seriously delayed. It is only when all these explorations, assays, and surveys have been made that it will be advisable to locate a tunnel or sluice for carrying off all the water and detritus, and subjecting it to amalgamation before

it is thrown upon the dump. No matter how favourable all these conditions may be, good business management will still be an indispensable requisite to success in such an undertaking. The ground contains large amounts of the precious metal, but it is scattered through a vast amount of barren material. To collect it, immense quantities of dirt must be very quickly and very skilfully worked. Any carelessness in the technical part will result in large losses, but want of business management will result in larger. Economical management is therefore an essential requisite to success. No hydraulic mine, however rich, could long contend against a cheap installation or extravagant management.

There are several conditions which are indispensable for working a hydraulic claim. Since the amount of gold contained in a cubic yard is exceedingly small, it is absolutely necessary to have the least amount of labour and the smallest expenditure of money possible, consistent with economic management. There must be a sufficient supply and head of water, not only to do the work of cutting down the banks, but also to carry off the material, and, most of all, there should be plenty of space for depositing the tailings after they have passed the sluices. One of the special reasons why hydraulic mining failed so often in its early history and for a number of years fell into disrepute, was either that sufficient "dump room" had not been provided, or no proper plan for carrying off the tails had been devised, or from want of care the facilities for doing so had become obstructed.

In order to justify the outlay it is necessary that the work should be permanent; and in establishing a mine of this character dump provision should always be made to carry off the tailings, not for months, but for years, in order that the capital invested may come back. When sufficient amount of fall below the outlet of the claim, or "dump," as it is called, cannot be obtained to get rid of the tails, it is useless to open it. As the supply of water is rarely sufficient to run any mine throughout the entire year, many of them will be closed for months together, and it will often be found that the spring rains have done for the company what it could not do for itself, that is, that the freshets have washed away the accumulation of tailings and carried them

further down the stream. The accumulations of such immense quantities of detritus have become a serious question between the farmers in the lowlands and the miners in the higher ones. and has resulted in what is known as the "débris litigation," which for ever restrains the miner from depositing the tails in the rivers, and has resulted in closing legitimate work in most of the hydraulic mines. The wheat grower claims that the miner is destroying the wealth of the State, while the latter asserts that without the privilege of washing his dirt into the valleys, California would no longer be either a gold or a grain-producing State, and that the value of agricultural lands is therefore of very secondary importance. A few years will show whether both the State and the farmer, by cutting off in this way by injunction the greatest supply of gold in the State, have not been the losers.

II. Survey of the Water Sheds.

The question then of working deep placers is not so much a question of the yield of gold, as of constant water supply. To insure this some of the most extensive constructions of modern times have been made by corporations calling themselves "ditch companies," who frequently work the claims, and do not sell the water, as this is found to be the most profitable investment for capital. These companies were at first organised to work their own claims, and then for the sale of their surplus water, then solely for the sale of water to miners only; but the water companies originally established to work gold mines now furnish water for irrigation, and for supplying towns, as well as for mining. The recent ditch company takes into its calculations the sale of water for any and every purpose for which it can be used. Three-quarters of the ditch companies, however, still use the water exclusively for their own supply, and do not sell it. Others sell their surplus and cut off the supply from the purchaser when the water gets low. These sales run from one to two million ten-hour inches per year. The water is sold in the various States to the farmers and others who use small quantities at the rate of 10 to 30 cents a ten-hour inch. To the hydraulic miners, who use large quantities of it, it is sold at rates varying with the quantity of from 4 to 15 cents. The ditch costs

generally from 3 to 3½ cents per inch for maintenance, v
including interest on the investment. The total cost of
tenance of the Milton ditch is 6½ cents, while with the Blo
ditches it is only 2.27 cents. The expenses are for repa
watchmen. The duty of the watchman is to patrol th
and report any accidents or infringements.

In California* the water rates vary very much. The
price is 3 cents for a twenty-four hour inch; the highest
25 cents for the same amount. The price is usually less
consumers, and more to those who, like the drift miners,
little or a variable quantity. The usual rates are 5, 7, 10
cents. If the water is used over again a less price is
One mine in Del Morte County sells its surplus water t
claims at $50 the season, and when its own works stop
whole of its water. In El Dorado County one ditch
sells the ten-hour inch to hydraulic mines at 6½ to 1
according to the quantity taken, to quartz mills at 20 ce
for purposes of irrigation at 25 cents. One large com
Tuolumne County sells its surplus water to two flour m
foundry, and for other power and for irrigation, as we
hydraulic mining, and publishes the following price list f
used in hydraulic mining:

One stream of 12 in.	...	per day ...
,, 15 ,,	...	,, ...
,, 18 ,,	...	,, ...
,, 21 ,,	...	,, ...
,, 24 ,,	...	,, ...
For larger amounts up to 48 in.	...	per inch ...
From 60 in. to 100 in.	...	,, ...

A company in Shasta County asks 5 cents for the
inch and 10 cents for the twenty-four hour inch. In th
Wood district of the Black Hills, Dakota Territory, t
mills pay $2 a week per stamp. In the Boisé Basin, I
rates are 10, 12½, 13, and 15 cents. In the case of on
30 cents per twenty-four hours is the price. The rates
not so much with the quantity as with the elevation.
parallel lines of ditches the lower one sells at 10 ce
the higher easily commands 15 cents. One company

* Report of U.S. Census, 1880, vol. xiii., p. 238.

ground to Chinese free and sells them the water, and makes an enormous profit in this way.

The rates in Montana are 10 cents for a ten-hour inch, 16 cents for a twenty-hour, and 15 cents for a twenty-four hour inch. In Baker, Oregon, the rates are 8, 10, 12, 12½, and 15 cents for a ten-hour inch. In a single case it is as low as 7½ cents for a twenty-four hour inch. One ditch owner uses a sliding scale, charging most to the most profitable claims. In Grant County 15 cents is charged for a twenty-four hour inch for drift mining, while only 6 cents is charged for hydraulic mining. In Georgia the rates are 12 to 12½ cents per inch. It may be said generally that the price of water depends more on the prosperity of the mines than it does on the abundance or scarcity of supply or the expense of carrying it. When the claims are new and rich, prices are charged which would be impossible when the country has been worked for a long time. If the miner finds that his work is costing too much, the ditch company must reduce its charges or let the ditch lie idle. Competition does not affect the prices of water to any extent. In most cases there is only one source of supply of water, and when there are several sources of supply the highest head, other things being equal, secures the best price. Before the stoppage of so many of the hydraulic mines it is estimated that nearly 1,000,000 twenty-four hour inches of water were sold yearly in that State.

The following Table gives the production of North Bloomfield, the water used, and the cost of water:*

Year.	Inches Used.	Yield per Inch.	Gross Yield.	Cost of Water per Inch.
		cents.	$	cents.
1867-70	Not known.	...	121,373.42	Not known.
1870-74	710,987	13.6	96,700.00	,,
1875	465,832	17.8	83,078.63	03.75
1876	786,200	25.5	200,366.54	03.10
1877	597,000	48.8	291,125.42	03.65
1878	796,449	39.1	311,276.70	02.60
1879	919,008	36.1	331,759.76	02.50
1880	744,600	31.8	236,935.00	02.79

Some idea may be had of the enormous amount of débris coming from the mine, when it is realised that in the year 1883 a

* Prod. Precious Metals in the U.S., 1881, p. 631.

volume of water of 862,660 in., miners' measurement, was at this mine every twenty-four hours.

The following Table shows the operations of the North Bl field and Milton Companies from 1877 to 1883, inclusive :*

Year.	Gold Product.	Water Sales.	Net Profi
	$	$	$
1877	670,774.57	6,753.67	364,045.4
1878	849,036.16	9,694.53	501,181.0
1879	794,517.90	9,090.84	462,821.4
1880	665,710.60	12,395.26	264,075.4
1881	637,682.06	24,233.03	325,226.6
1882	802,190.40	5,134.56	416,208.6
1883	738,985.97	7,031.64	400,089.7

NOTE.—The main reservoir of the Milton Company broke June 18, 18 and the company was deprived of the benefits of half a year's run, ther materially decreasing the profits. The water stored in the reservoir ample to continue operations for the balance of the season.

The number of companies who sell their water is report to be :†

State or Territory.	Number of Ditch Lines Reported.	Number of Lines which Sell Water in Part or Wholly.	Number of Lin which Supply only Owners' Mines.
Alabama . . .	1	...	1
Arizona . . .	2	...	2
California. . .	40	14	26
Colorado . . .	6	...	6
Dakota . . .	6	1	5
Georgia . . .	8	3	5
Idaho . . .	17	10	7
Montana . . .	11	3	8
Oregon . . .	52	9	43
Total . .	143	40	103

It will thus be seen that hydraulic mining divides itself into two principal departments, water supply and hydraulic mining proper. For hydraulic mining abundant and cheap water is as indispensable a requisite as the possession of a gravel deposit containing the precious metal. Drift mines require less water, but it is essential to their successful operation. Both water supply and mining were included in the works of the same claims in the early days, but the question of water supply has become

* Prod. Precious Metals in the U.S., 1883, p. 714.

† Report U.S. Census, 1880, vol. xiii., p. 238.

one of such magnitude that the tendency, except with companies who have very large capital, is to separate the two, so that the water or ditch company has no interest in the mining, but obtains its revenue solely from the sale of its water. The conditions under which this is done are, for the most part, peculiar to the extreme Western States and Territories of the United States, where this industry originated, and although it has suffered immensely from the recent decisions of the Courts against it, it seems now more than likely that not only will this method of mining be in some way revived, but that the immense impetus which it has given to hydraulic engineering, will make its water power, which, since the injunction, has been almost unused, available for nearly all the industries of the State. In some cases the mines for which this large supply of water was made have become either exhausted or have for some other reason been abandoned. It now seems likely that the power which they are capable of producing will be used for manufacturing purposes, and if eventually the Courts decide that the mining companies can resume work, there will be a much larger amount of water used than ever before. This injunction of the Courts has developed the application of water power by means of hurdy-gurdy wheels to other purposes than mining much more rapidly than would otherwise have been possible.

The water is sold at so much the miners' inch. A miners' inch is the quantity of still water which will flow through a given aperture under a given pressure in a given time. It is an arbitrary measure which was adopted in the early days of mining by the ditch companies who had water to sell, each one fixing the standard best suited to its own convenience, and afterwards established by the law of each miners' camp without reference to what had been done elsewhere. It consequently varies in different districts, and sometimes in the same district, according to the size of the opening, the pressure of the water, and the way it is discharged, so that without knowing the locality it is impossible to state what the miners' inch is. Many attempts were made in the early days to fix the miners' inch, but no general law was enacted, and in most cases no law at all and no guide except the custom of various districts. In the

year 1866-67 a bill was introduced into the Legislature of the State of California, making the miners' inch to be the same as the machinists' inch, or 2½ cubic feet, or 145.86 lb. of water per minute. This bill was referred to a committee, who reported a substitute defining a miners' inch to be "that quantity of water which would pass through an orifice 1 in. square in a plank 1 in. thick with a pressure of 7 in. of water measured from the centre of the orifice, provided the water shall be delivered from a box in which the water has no motion except that caused by the flow of water to be delivered from that particular box." The consideration of both bills was indefinitely postponed* on the ground that the miners and the ditch companies had better agree on the measurements as well as on the prices for water. It would seem that under such conditions great injustice would be done to those purchasing the water, but the price is more arbitrary than the supply, so that the cost of the water in the annual statement of the different mines varies more with the quality of the ground than it does with the two other variables, and no practical loss has arisen from it. However, for the purpose of comparison between the different districts it would undoubtedly be a very decided advantage to have a uniform standard of measuring the water and the same mode of calculating it adopted throughout the country.

The Table† on the next page gives the various definitions of the miners' inch in eleven different States and Territories of the United States.

The definition of the inch most largely used is the amount of water discharged through an opening 1 in. square in a plank 2 in. thick, with still water standing at a pressure of 6 in. above the opening, or 7 in. over its centre. The discharge is sometimes flush with the bottom of the reservoir and sometimes raised above it. The thickness of the plank through which the discharge is made is 1 in., 1½ in., and 3 in. The lower front end of this discharge is usually chamfered. The height of the water over the orifice, which is called the head, is, however, not a fixed quantity. It is sometimes counted from the centre of

* Mineral Resources of the U.S., 1868, p. 184.

† Report of the U.S. Census, 1880, vol. xiii., p. 240.

Definition Locally Accepted.	Total Number of Definitions.	Arizona.	California.	Colorado.	Dakota.	Georgia.	Idaho.	Montana.	North Carolina.	Oregon.	South Carolina.	Utah.
Stream of water passing through—												
"A square inch orifice under 6-in. head," thickness of partition and point from which pressure is measured, whether top, centre, or bottom of orifice, not stated	97	1	20	10	11	5	1	11	...	38		
"A square inch orifice under 6-in. head above top," thickness of partition not stated	1	...	...	...	...	...	...	...	...	...	...	1
"A square inch orifice under 6-in. head, measured from centre; 2-in. plank"	3	...	3									
"A square inch orifice under 6-in. head," point from which pressure is measured not stated; "2-in. plank"	7	...	7									
"A square inch orifice under a head varying from 4 in. to 6 in."	2	...	2									
"A 1-in. orifice under 4-in. head"	7	...	3	...	...	4						
"An orifice 2 in. high under 5-in. head," delivering 2 miners' inches per inch in width of orifice	1	...	1									
"An orifice 2 in. square under 6-in. head from centre," =4 miners' inches.	1	...	...	...	...	...	...	...	...	1		
"An orifice 3 in. high under 3-in. head," delivering 3 miners' inches per inch in width	9	...	...	...	...	...	...	...	...	9		
"An orifice 3 in. high under 6-in. head," delivering 3 miners' inches per inch in width	1	...	...	...	...	...	...	...	...	1		
"An orifice 4 in. high under 4-in. head," delivering 4 miners' inches per inch in width	2	...	1	...	...	...	...	...	...	1		
"An orifice 4 in. high under 6-in. head, above top of aperture," delivering 4 miners' inches per inch in width	1	...	...	...	...	...	1					
"An orifice 6 in. high under 3-in. head above top, shingle partition," delivering 6 miners' inches per inch in width	1	...	...	...	...	...	1					
A delivery of—												
"10 gallons per minute"	2	...	...	...	...	...	...	1	1			
"675 gallons per hour"	1	1										
"1.5 cubic feet per minute"	1	...	...	...	...	...	...	1				
"10 gallons per minute; or, water flowing through an orifice 1 in. square in a thin vertical partition, under 5-in. head."	1	...	...	...	...	...	...	...	...	...	1	
Running inch: in cross section of ditch or flume, without pressure	8	...	8									
Total number by States and Territories	146	2	45	10	11	9	3	13	1	50	1	1

the orifice and sometimes from the top of it. It usually varies from 4 in. to 6 in.; with the South Yuba Ditch Company it is 10 in., and in exceptional cases it may be as high as 12 in. In some cases it is as low as 2 in. The quantity of water which will in twenty-four hours flow from an aperture 1 in. square through a 2-in. plank with still water standing 6 in. above the orifice was determined by Mr. J. D. Hague as 1.5625 cubic feet per minute, 93.75 cubic feet per hour, or 2250 cubic feet in twenty-four hours. Other observers have made it 2274 cubic feet in twenty-four hours, equal to very nearly 17,000 gallons in twenty-four hours, others 700 gallons per hour, or 16,800 gallons in twenty-four hours, and this is the generally received amount.

The following Table* shows the variations in some of the miners' inches:

Name of Mine.	Height of Opening.	Length of Opening.	Pressure over Centre of Opening	Quantity Discharged by One Square Inch per Minute.	Quantity Discharged by One Square Inch in Twenty-four Hours.*
	in.	in.	in.	cub. ft.	cub. ft.
Smartsville . .	4†	250‡	9	1.76	2534.4
North Bloomfield .	2	50	7	1.57	2267
Eureka . . .	2	48	6	1.39	2001.6
South Yuba Canal .	2	50	6	1.44	2074

* The day is usually reckoned as eleven hours.

† The bottom of the aperture is on a level with the bottom of the box.

‡ An opening 250 in. long and 4 in. wide will discharge 1000 Smartsville inches.

The opening through which the water flows is never less than 1 in. high and is sometimes 1 ft. Its length varies from a few inches to several feet. The plank through which it discharges is usually 1 in., 1½ in., 2 in., or 3 in. thick. In some cases the edges of the opening are square, in others they are square and chamfered. The Smartsville inch is equal to nearly 19,000 gallons in twenty-four hours. The La Grange inch is equal to 2159 cubic feet in twenty-four hours. To determine the value of the North Bloomfield inch the water was made to discharge through a 3-in. plank. The bottom of the opening was above the bottom of the tank holding the water, and was chamfered

* Trans. Inst. Min. Eng., vol. vi., p. 8.

off 1 in. from the outside, so that the outside opening was 4 in. high. The value of this inch is given in the preceding Table as determined. The South Yuba Canal Company discharges through a 2-in. opening in a 1½-in. plank under a pressure of 6 in. The Milton and La Grange Companies discharge through an opening 2 in. wide and 50 in. long in a 3-in. plank with a head of 7 in. The Eureka Lake and Canal Company use an opening 2 in. high, 10 in. long, under a head of 6 in. This gives exactly 20 miners' inches of their measurement. Occasionally the pressure is made 3 in. to 4 in., and very exceptionally it is discharged without any head.

In order to ascertain exactly what quantity of water is being used it is usual to specify the number of seconds, minutes, or hours that the water is to flow. These are then called second, minute, or hour inches. Ten hours is usually the shortest time. Eleven to twelve hours are generally used. Twenty-four hours is quite common in some districts. These are then called ten, eleven, and twenty-four hour inches. The opening through which the water discharges, however, is never made 1 in. square, for this would give an insufficient supply. It is always longer or higher, which would necessarily increase the quantity of water that would issue from each square inch. The height of the water will also affect the quantity which will be delivered in a given time. This height is in some cases measured from the top, and sometimes from the centre of the orifice from which the water issues. The form of the opening, its height, and in a certain degree the distance between the lowest point and the bottom of the measuring box, the thickness of the plank through which the water flows and the nature of the approaches, the character of the fall on the delivery side of the aperture, will also affect the quantity of water to be delivered. These conditions all vary in different localities, and the amount of water which is delivered from any particular inch varies accordingly. The laws given below,* regulating the quantity of fluids discharged, ought in every case to be applied in determining any flow.

1. That the quantities of a fluid discharged in equal times by the same apertures from the same head are nearly as the areas of the apertures.

* Engineers' and Machinists' Pocket-Book, C. Haswell, 1879, page 359.

2. That the quantities of a fluid discharged in equal times by the same apertures, under different heads, are nearly as the square roots of the corresponding heights of the fluid above the surface of the apertures.

3. That the quantities of a fluid discharged during the same time by different apertures, under different heights of the fluid, are to one another in the compound ratio of the areas of the apertures and of the square roots of the heights of the fluid above the centre of the aperture.

4. That, on account of the friction, small-lipped or thin orifices discharge proportionally more fluid than those which are larger and of similar figure under the same height of fluid.

In order to ascertain what in actual practice the value of the inch was, Mr. H. Smith, in 1874, undertook a series of observations at Columbia Hill, at an elevation of 2900 ft. above the level of the sea, and others were made in 1876 by Mr. A. J. Bowie, at La Grange, Stanislaus County, at an elevation of 216 ft. above the level of the sea. They were both made from a 2-in. opening through a 3-in. plank chamfered to 2 in., with a 7-in. head. The observations are given in the Table below :*

—	H. Smith, 1874.	A. J. Bowie, 1876.
	cubic feet.	cubic feet.
One miners' inch discharges in one second .	.02624	.02499
,, ,, ,, ,, minute .	1.5744	1.4994
,, ,, ,, ,, hour .	94.4640	89.964
,, ,, ,, 24 hours .	2267.1360	2159.146
Ratio of actual to theoretical discharge .	61.60 p. ct.	59.05 p. ct.

The State of California has, to settle legal claims, made a statute inch. It is found in Section 1415 of the Code of the State of California, which provides in these words: "That he (the locator) claims the water there flowing to the extent of (giving the number) inches measured under a 4-in. pressure."

On these data the value of the statutory miners' inch is as follows :†

	Cubic Feet.
For one second, "second inch"	0.02
,, minute, "minute inch"	1.20
,, hour, "hour inch"	72.00
For twenty-four hours, "twenty-four hour inch" ...	1728.00

If a cubic foot be divided by the flow in one second, there will result the number of miners' inches equal to the discharge of

* Trans. Am. Inst. Min. Eng., vol. vi., p. 51.

† Prod. Precious Metals in the U.S., 1884, p. 559.

1 cubic foot per second. It is thus found that 1 cubic foot per second is equal to 50 statute inches.

In order to get a regular quantity of water the discharge opening is made more than 1 in. high. Thus an aperture 1 in. wide and 2 in. high would discharge 2 in., one 3 in. high 3 in., and so on. If the opening is wider, say 24 in., then an aperture 2 in. high and 24 in. wide would discharge 48 in., and so on. In order to be able to measure the water as it flows it is discharged from a gauge box. This is usually made of pine boards from 2 in. to 3 in. thick. It varies from 2 ft. to 4 ft. or more in width according to the supply that is needed. The depth usually varies from 12 in. to 24 in. or more in height. The discharge opening is made by nailing to the bottom of the box a strip of wood from 1 in. to 3 in. in height, whose outside edge is chamfered down for 1 in. or more so as to reduce the width of the strip over which the water flows.

The width of the aperture is determined by a gate which is adjustable and can be fixed at a given distance above the bottom of the orifice. The height of the water from the centre of the orifice is determined by a gauge so as to ascertain the pressure. If the orifice is 2 in. high and 50 in. wide with a pressure of 6 in. above the centre, each inch in width will discharge two miners' inches, or 100 for the whole width of the box under a 6-in. pressure. The exact number of inches to be used at any time is determined by the use of a gate, which, when fixed at a determined height, is held in place by the force of the water. This gate closes a determined width of the orifice. This box may be placed anywhere, and as all the water in the ditch must pass over it, it is a very simple contrivance for determining the amount of it. The following Table gives the various values of the miners' inch in different districts:*

——	Cubic Feet per Minute.	Cubic Feet in Twenty-four Hours.	Gallons in Twenty-four Hours.†
Smartsville	1.76	2534.4	18,958.63
Eureka Ditch	1.39	2001.6	14,972.98
North Bloomfield	1.5744	2267.136	16,959.36
Opening 1 in. sq. with a 6-in. head	1.99	2274	16,512.50

* Prod. Gold and Silver in the U.S., 1881, p. 645.

† 1 U.S. gallon = 8.34 lb.

It will be seen by the above Table that there is considerable variation in these determinations; 17,000 gallons in twenty-four hours is not far from a mean. The quantity of water sold to or used by the mining company is called a "head." The "head" varies from a few hundred up to several thousand inches. Few companies purchase as much as 4000 in., though several who own ditches use more. Very few companies run a "full head" all the year round, but diminish it as the dry season comes round.

Generally the average delivery of the miners' inch is assumed to be 700 gallons per hour, which corresponds to 0.1944+gallons per second, and 11.666+ gallons per minute, and 16,800 gallons for twenty-four hours. A gallon of water is equal to 231 cubic inches, and weighs 8.3388 lb.; 1728 cubic inches, or 1 cubic foot, contains 7.48052 gallons, and weighs 62.37856 lb.

At the South Yuba Canal, Nevada City, a miners' inch is the quantity of water that will pass a 2-in. hole in a 1½-in. plank with 6 in. pressure. The velocity of the water under 6 in. pressure will be 3.9 in. per second. The usual head of water sold to any one company by the South Yuba Ditch Company is 500 in. measured in a box having a discharge 4 in. by 195 in. with an opening 9 in. from the centre of discharge. Mr. Randall* has given the quantity of water which will flow under a given pressure from a 1-in. opening.

FLOW OF WATER THROUGH VERTICAL RECTANGULAR OPENINGS.

Head on Centre.	Flow per Second through One Square Inch Opening.	Minute Inch Flow.	Hour Inch Flow.	Twenty-four Hour Inch Flow.	One Cubic Foot Flow per Second.
in.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	miners' in.
3	.0169	1.014	60.84	1460	59.18
4	.0195	1.173	70.42	1690	51.13
5	.0218	1.309	78.54	1885	45.86
6	.0239	1.434	86.00	2064	41.85
7	.0258	1.548	92.92	2230	38.74
8	.0276	1.655	99.33	2384	36.24
9	.0293	1.755	105.33	2528	34.17
10	.0308	1.851	111.04	2665	32.42
11	.0323	1.941	116.46	2795	30.91
12	.0338	2.028	121.67	2920	29.59

* Prod. Gold and Silver in the U.S., 1884, p. 570.

The duty of a miners' inch of water is the quantity of dirt it can carry through the sluice. This duty is very variable, since the miners' inch is not a fixed quantity, as it varies from 1.2 to 1.78 cubic feet per second in different sections. Part of the force of the water is expended in cutting down the bank, but its principal object is to get rid of the débris. The duty will vary with the nature of the gravel, the condition of the ground, the force of the water, and the size and grade of the sluice. It will vary according to the nature of the gravel from 2 to 4½ cubic yards. It is always less on cement, and may be as low as 2 to 3½ for that which has been blasted. The duty on the top gravel may be nearly twice or even three times as much, and varies between 5 and 10 cubic yards. When the gravel is light and when the nozzles are 6 in. to 8 in. in diameter, with a pressure of 150 ft. to 300 ft., they will wash from 5 to 7 cubic yards of gravel weighing 1.8 tons per yard. The North Bloomfield Company, No. 8 claim, in 1876, ran 342 days, and washed from a bank 260 ft. high 2,919,700 cubic yards. The duty of a twenty-four hour inch on cement in five mines is given below:*

	Cubic Yards.
North Bloomfield	3.25
Milton	2.4
Excelsior	2
Gold River	3.5
Dutch Flat	1.4

On the upper gravel the duty is:

Gold Run	3.9
Yuba River Mines	3.5
Bear River Mines	3
American River Mines	4.5
Cedar Claim, Dutch Flat	7.5

The duty of the miners' inch was determined by observation in 1882 at several leading mines in California as follows:

Name.	Water.	Earth moved.
	Miners' inches.	Cubic yards.
North Bloomfield Mine	2,800	11,200
Milton	3,300	11,400
Omega	1,500	7,500
Conly and Gowell	1,700	10,200
Excelsior	1,600	6,400
Chadwick and Campbell	1,200	14,400
	12,100	61,100

* Prod. Precious Metals in the U.S., 1884, p. 567.

This includes working both the top and bottom gravels which were not separated in these observations, and shows an average duty of 5 cubic yards per twenty-four hour inch. An instance is cited where the duty was as high as 19 cubic yards. It has been found that the duty varies nearly as the square of the inclination of the sluice. A sluice with a grade of 12 in. in 12 ft. will carry four times as much as one with only six. With a grade of 12 in. boulders weighing from 200 lb. to 500 lb. pass easily and those of from 1000 lb. to 2000 lb. pass slowly through the sluice.* In the cement the excavating power of the water is rarely as great as the transporting power, but in the upper gravels it is frequently greater. In this case the transporting power becomes the duty, which may be increased by increasing the grade. In the various suits which have been recently carried on the estimates have differed as much as 33 per cent. of the total amount actually washed. This is accounted for, first, by the great variety in the miners' inch, secondly, by the uncertainty that always exists of measuring the ground from witnesses after the earth has been washed away.

III. The Construction of Reservoirs.

A large and constant supply of water has of late years become such a necessity in the placer districts that the greatest engineering skill of California has been concentrated upon bringing the water to the point where it is to be used. There is no natural deficiency of water in these regions, but it is not available when and where it is wanted, either because the gold-bearing deposits are too high or away from the natural drainage of the country. The average yearly rainfall at Sacramento for thirty-five years was 19½ in.† The average for ten years at the summit of the Sierra Nevadas has been 74.6 in., and the average for twenty years at Nevada City at an elevation of 3000 ft. has been 54.43 in. In California and in Oregon the altitudes of these placer deposits are comparatively low. The mean elevation of these deposits for the United States‡ has been found to be 3447 ft. above the level of the sea. From the 15th of April to

* Prod. of Precious Metals in the U.S., 1883, p. 710.

† Report of the U.S. Census of 1880, vol. xiii., p. 215.

‡ See page 82.

the 15th of November there is but little rain in California. By about the 15th of July most of the small streams have dried up altogether, and the large ones are so low as not to furnish sufficient water for mining, so that the claims must either stop working or construct reservoirs to store the water during the dry season.

The source of supply of the water will be necessarily from springs, lakes, creeks, and rivers, yet these run dry in the summer, or are subject to freshets from the melting snow in the spring, and from the heavy rains in the Fall, and are consequently very variable in the quantity of water that they can supply for any given time. Most of the sources of supply are above the winter snow line, which makes the miners' supply uncertain, causing a want of water during the summer season and a great excess of it in the spring. There are but few who can avail themselves of the natural storage, such as lakes, so that water reservoirs become a great necessity, and building them a matter of great expense as well as engineering skill. The first requisite in the arrangement of hydraulic plant is that it should have a full supply of water during all seasons of the year. If money can accomplish this object no expense should be spared, as the return for capital invested in such works has always been large. There is no natural deficiency of water, but it is never in great abundance where it is wanted for working large claims, so that those companies who formerly depended on ditches alone are obliged to lie idle during the summer months. If, therefore, by any outlay, or by engineering skill, a constant supply can be secured, any expense will be justified. The summer supply is more important than it would at first seem, since under the great heat of the sun the water becomes almost tepid, the days are long, the workmen less exposed to the inclemencies of the weather, and on account of the heat the quicksilver acts much more readily, and the yield of gold will be consequently much higher for this reason alone in the summer than in the winter. The natural supply of water is very variable, being constantly subject to maxima and minima, which represent long droughts or serious floods; it is, therefore, necessary to build dams of the strongest construction, to retain the water so as to have a constant supply, and distributing reservoirs near the places where it is to be used, to supply the

miles of ditches and flumes which carry the water to the various claims. These constructions have not only put the best engineering ability of California to the severest test, but they have in many cases involved the outlay of many hundreds of thousands of dollars, and in some few cases of over a million. Some of them take the drainage from a district of from five to forty square miles. The construction has demanded a thorough revision of the science of hydraulic engineering and the possibility of water supply, and under any circumstances seems to insure a magnificent future for the manufacturing industries of California.

The following Table* gives the capacity in cubic feet of the principal reservoirs on the Feather, Bear, Yuba, and American rivers:

Name.	Capacity in Cubic Feet.
North Bloomfield	1,050,000,000
Milton	650,000,000
Eureka Lake	1,130,000,000
South Yuba	1,800,000,000
Omega and Blue Point	300,000,000
Spring Valley	300,000,000
California	600,000,000
El Dorado	1,070,000,000
Other small reservoirs on the Feather, Yuba, and American rivers	700,000,000
Total storage	7,600,000,000

The following Table gives the high-water area, the height, and the capacity of the principal reservoirs† in Nevada County, supplying water to the North Bloomfield Mining Company, the Milton Mining and Water Company, and the Eureka Lake and Yuba Canal Companies Consolidated:

Name of Reservoir.	High-water Area.	Height of Dam.	Capacity.
	acres.	ft.	cubic feet.
Saw Mill Flat . .	806.0	39	70,000,000
English, now broken .	363.0	131	930,000,000
Bowman . . .	490.0	96	796,700,000
Eureka Lake . .	337.0	64	640,000,000
Faucherie . . .	90.0	21	58,800,000
Weaver Lake . .	83.5	22	57,800,000
Rudyard . . .	...	100	535,000,000
French Reservoir .	...	...	661,000,000

* Report of U.S. Census, 1880, vol. xiii., p. 235.

† Prod. Gold and Silver in the U.S., 1883, p. 710.

The North Bloomfield Company, the Milton Company, and the Eureka Lake Company have a catchment area of 68.6 square miles. They have eleven principal reservoirs, covering at high water from 10 to 487½ acres, which have a capacity of from 2½ to 796.7 million cubic feet. The total of all the reservoirs is over 11,600 acres and the total capacity is 2195 million cubic feet. The North Bloomfield Company use 15 billion gallons per year, which is nearly three times as much as the Spring Valley Company supply to the City of San Francisco. Their reservoirs cost about $242,073, and the ditches about $466,767.* The largest tract of land belonging to any single system of reservoirs and ditches is that of the Trinity River Canal and Hydraulic Mining Company, which comprises an area of 1700 acres.

The total storage capacity of the mining reservoirs of the State of California is approximately as follows:

	Cubic Feet.
North Bloomfield Gold Mining Company's reservoirs hold	1,050,000,000
Milton Mining and Water Company, now broken ...	650,000,000
Eureka Lake and Yuba Canal Companies Consolidated	1,130,000,000
South Yuba Water Company	1,800,000,000
Omega and Blue Tent Mining Company	300,000,000
Spring Valley Gold Mining Company	300,000,000
California Water Company	600,000,000
El Dorado Company	1,070,000,000
Small reservoirs on Feather, Yuba, and Bear rivers	700,000,000
Estimates of reservoirs in Amador and Calaveras Counties...	400,000,000
Total	8,000,000,000

The water of the Spring Valley Company is drawn from nine reservoirs which occupy an area of 522 acres, have 94 miles of ditches and 20,940 ft. of 30-in. wrought-iron pipe. The most costly of the single reservoirs is the one of the South Yuba, at Nevada City, Nevada County, California, which cost $875,000. The next is the Spring Valley, which cost $120,000. Most of these dams, however, have cost from a few hundred up to from $2000 to $5000. The cost of the construction of the North Bloomfield is annexed.

* Prod. Gold and Silver in the U.S., 1881, p. 63.

Fifty-three miles of ditch	... $426,000
Bowman reservoir...	... 135,000
South Lake dam	... 5,000
Waldron, Blair, and Ruff reservoirs	... 18,000
Total	... $584,000

The Bowman dam has in it 56,000 cubic yards of material, which weigh about 86,000 tons. The outer walls of the dam are of dressed stone; the catchment basin covers an area of twenty-nine square miles. The average rainfall is about 78 in., three-quarters of which are caught in the reservoirs. The total cost of the whole reservoir, composed of a series of dams, has been $214,400.

The Eureka Lake and Yuba Canal Company has constructed reservoirs by damming the outlets to several small lakes, making them capable of holding 820,000,000 cubic feet. The Meadow Lake and Fordice Valley reservoirs have a water catchment of over 100 square miles. All the large companies have besides the main reservoirs distributing reservoirs to receive and hold the water for the ditches.

The cost of the dams and reservoirs with their ditches is often increased by accidents; thus, at North Bloomfield, the Bowman reservoir was burned down and cost $135,000 to rebuild. The whole object of the construction of these dams is to take in not only the running water supply but to catch all the waters from rains and freshets and snows melting in the mountains, and store it in the spring and summer months for use during the season when no rain falls. Many of the large ditch companies have established a complete system of storage reservoirs, single companies sometimes storing in several of them over 1,000,000,000 cubic feet of water at a cost for water alone of $250,000.

The dams are generally constructed at some point where the stream narrows, and has heavy rocks on both sides. They are made of stone or of wood, that material being used which is nearest. When built of stone some of them are made of cut granite and some of the heavy slates of the district; they are put together without mortar, but are keyed. At Eureka the front wall is built of cut granite, the largest stones of which are about one-third of a cubic yard; at Meadow Lake they use pieces

of schistose rock weighing from 150 lb. to 180 lb. The stonework of the Eureka dam, built in 1857, forms about half the volume of the dam, the rest being composed of puddled clay and earth. The inside surface is inclined at 45 deg., and is covered with plank 3 in. thick, 6 in. to 10 in. wide, and 4 ft. 10 in. to 10 ft. long, fastened to beams 10 in. square, of which there are 2700 running feet. The planks are caulked, and have a superficial surface of 1050 square yards.

Height of dam	69 ft.
Volume of stones with wood	300,000 cub. ft.
Weight of the material	19,000 tons
Pressure of the water	6,300 ,,

The outlet is built of cut granite blocks. The water is carried out in a flume 3 ft. 10 in. wide and 1 ft. 10 in. high, built of 2½ in plank, supported on beams 12 in. square. The whole surface covered by water is from one-third to one-half a square mile, and the capacity of the reservoir is 450 million gallons.

Wooden dams are made by cutting down the trees, running them from shore to shore, 6 ft. to 8 ft. apart for a width of 40 ft. to 50 ft., then placing other trees at right angles to these, about the same distance apart, building up in this way a sort of crib. Where the trees cross they are fastened together with bolts. The interval between the trees is filled up with gravel and stones. The trees that lie parallel with the stream are always put in with part of their branches on them, the branches heading up stream. The inside of these dams is usually covered over with planking, which generally makes the face of the inside watertight. The arrangement should be made if possible so that the gate should be constructed in solid rock, and will be all the better if a ledge is selected through which a tunnel can be made for this purpose. Some of these dams are 75 ft., 90 ft., and 100 ft. high. The dam is constructed at low water, and the ditch commenced immediately. The arrangements should be made so as to have the ditch on a lumber line, so that the timber which will be needed for it and the flumes can be floated down the ditch, and in this way save transportation. A portable sawmill is an indispensable requisite in the building of dams and flumes.

IV. The Construction of the Ditch.

The necessity of having some supply of water other than that which could be obtained from natural sources was shown as early as 1850, when reservoirs were constructed and ditches built to bring to the miners a "tom-head" of water, which was sold at profitable rates. As a business enterprise the sale of water was almost universally successful. When hydraulic mining passed into the state into which almost all new enterprises pass, which, founded on great principles, become exaggerated, ditch properties fell into disrepute, making such poor returns that they were often sold to satisfy judgments, or abandoned altogether as the cheapest way of disposing of the property. This was caused either by the construction of more ditches than the mining properties in the vicinity justified, or by the mistakes which the want of experience and too great haste to get returns caused the early ditch builders to make. These were faulty location of the ditches, the erection of very high flumes, either too slightly built, so that they were soon blown over, rotted or burned down, or of too costly ones, which, while they stood the wind, were proof neither against rot nor fire. Experience once gained, the short-lived, costly structures were replaced by iron pipes or well designed and constructed ditches in earthen banks, until now these water conduits have increased to such a size and number that many of them, with their great reservoirs and miles of ditches and flumes, both from the boldness of their design, the originality of their construction and great extent, are to be placed among the monuments of the great engineering skill of this century.

From the reservoirs the water is carried in ditches, flumes, and pipes to the place where the water is to be used. The requirements of hydraulic mining have made these ditches one of the greatest sources of power in the State of California. They not only now supply what miners are using but furnish the power for quartz mills and other mechanical uses, as well as water for irrigation. There are, according to the census of 1880, 10,783 miles of ditches in the United States, which, with the reservoirs, flumes, and pipes, represent a capital of $27,000,000 invested in this kind of enterprise alone.

The average date of the original construction of the lines now

in operation was 1867, which may be taken as the time of greatest activity in ditch building. Many of these ditches, however, have been enlarged or improved since that time to such an extent as to amount to a virtual reconstruction. A few ditches in California and Oregon, still in useful service, date back to 1850. Of existing ditches in Montana the oldest were built in 1861, and in Idaho the oldest date from 1863; but the average life or period of useful duration from the time of first construction up to the close of 1880 is thirteen years. For California, owing to the earlier inception of hydraulic mining, the average is still higher, or eighteen years.

The selection of the site of the ditch requires the greatest engineering ability and judgment, since until the stream itself has tightened all the pores of the ground, it will always give a great deal of trouble at the best; and if sufficient care has not been taken in its construction, the repairs may, within a few years, amount to more than the ditch originally cost; and even then the defects of the original construction may not have been remedied. It is very poor policy to economise on the first construction, for repairs will rarely make a good ditch out of one which was poorly built at first.

In the line of the ditch, tunnels should not be considered to be an obstacle, and the ditch itself will be a succession of ditches, properly speaking, of tunnels, of flumes, and of iron pipes. In all cases the line of snow should be avoided, or if this is not possible, those points where the snow is likely to accumulate. But it must always be remembered that the higher the elevation of the ditch, other things being equal, the greater the territory it will command and the greater the pressure at which it will be able to deliver the water. Great pressure is sometimes almost as necessary as large quantity of water. If the ditch is run too low these advantages are lost. If it must run above the snow line, provision must be made for protecting it either by snow sheds or by covering it thickly with boughs on which the light snow rests, and over which the heavy snows arch and support themselves. In selecting the ground through which the ditch is to pass, care must be taken to choose that which is solid, and to avoid as far as possible all material likely to produce

slides. It should not be built upon very steep inclines, but should always be placed so as to be far enough away from the slope of the hill, to have a strong body of earth on the outside bank so as to make it perfectly secure. It may happen that the nature of the ground will not admit of the outside bank being of earth. In such a case it will generally be best to secure it with masonry, or if that would be too expensive, to cut off the ditch and run a flume, though this would be the most expensive in the long run.

In selecting the site for the ditch, one of the first things to be observed is that it shall have a full supply of water at all seasons of the year, for which purpose it is well to make use of the streams on the route of the ditch, and so to secure them that they may be made use of if water is needed, or, if the streams are dangerous, that they may be carried either over or underneath it so as to avoid that element of danger. Another capital point to be considered for the ditch is, that it shall arrive at the point where the water is to be used at the highest possible elevation. This has the double object of not only supplying the mines which are in view at the time the ditch is constructed with water at a great hydrostatic pressure, but also looking forward to the possibility of other mines being established elsewhere to which the ditch company will in the future supply water.

Whenever there are projections of rock which would cause difficulty if the ditch were to be carried round them, they should be tunnelled. It may be necessary to do this either to shorten the route or to secure a passage on a proper level. Usually these tunnels will be short, but they sometimes are as long as 1000 ft. to 2000 ft., or even more.

The whole route of the ditch should, therefore, be carefully surveyed with a view to keep it as far as possible in earthen banks. A grade of at least 6 ft. to 10 ft. per mile ought to be established, as this has been found by experience in California to be the best on the whole, and the one which, while giving the proper velocity, will endanger the banks of the ditch the least. Ditches with a grade of 15 ft. to 40 ft. to the mile, delivering 80 cubic feet per second, have been successfully operated, but this is exceptional. The

grade once settled upon, should be adhered to throughout. The ditch is generally built in sections, which must be connected as rapidly as possible, to be used for the transportation of material. The grade of the larger ditches ranges from 9 ft. to 20 ft. per mile. In smaller ditches it runs from 5 ft. to 25 ft. per mile. The average is about 13 ft., or $\frac{3}{8}$ in. to the rod, which gives sufficient delivery without making the current so strong as to endanger the banks from too heavy a wash. The most recent practice is to increase the grade up to about the point at which the banks will resist the erosion. In a single case in Oregon the grade has been as high as 45 ft. In two instances in the same State, and in a single instance in California, the grade is 40 ft. In two other cases in Oregon the grades are above 30 ft., but by far the larger part are below 25 ft. The higher the grade the less the depth will have to be. If the ditches are built in light soil the grades are necessarily low; but where the ground is firm and tenacious the present tendency is to increase the grade, thus giving a more rapid current, and consequently greater capacity in proportion to the cross-section of the ditch. It is always preferable to make the ditch deep rather than wide on account of the evaporation. A swift current through a narrow ditch will furnish the same amount of water as a slow current through a wide one, and has the additional advantage of keeping the ditch clear of snow, ice, or any other encumbrance likely to clog it. A ditch with a slow current would certainly become clogged by the falling leaves in autumn and by the snow in winter. This rule, however, cannot always be followed if the ground above the rock is shallow. It may be that by coming too near the bed-rock there will be greater leakage than evaporation, in which case a shallow ditch must be made. It is evident that the less the time occupied in transit the less is the proportionate amount of leakage and evaporation; and, besides this, it is found that narrow, deep, and comparatively steep ditches are less liable to damage by storms, and are more readily repaired. Grades of 16 ft. to 20 ft. are now recommended by some engineers for lines cut in good ground.

The gradient is expressed either in inches and fractions of

inches per rod or in feet per mile, the latter mode is generally adopted. For convenient reference the following conversion Table, giving the equivalents in the two systems, is annexed:

GRADE EQUIVALENTS.*

Inches per rod .	$\frac{1}{8}$	$\frac{1}{4}$	$\frac{3}{8}$	$\frac{1}{2}$	$\frac{5}{8}$	$\frac{3}{4}$	$\frac{7}{8}$	1
Feet per mile .	$3\frac{1}{3}$	$6\frac{2}{3}$	10	$13\frac{1}{3}$	$16\frac{2}{3}$	20	$23\frac{1}{3}$	$26\frac{2}{3}$
Inches per rod .	$1\frac{1}{8}$	$1\frac{1}{4}$	$1\frac{3}{8}$	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{7}{8}$	2
Feet per mile .	30	$33\frac{1}{3}$	$36\frac{2}{3}$	40	$43\frac{1}{3}$	$46\frac{2}{3}$	50	$53\frac{1}{3}$

The velocity of water is generally from one and a half miles per hour in the lines of least grade up to five miles per hour in the steepest ditches. The average current is slightly over three miles per hour. The cross-section of the ditch is usually trapezoidal, the width at the top being considerably greater than that at the bottom, though the newer ditches are cut with less slope to the banks than was formerly the rule, as well as deeper in proportion to the width.

There is the greatest possible variation in the dimensions of these ditches as well as their grade. It will be noticed that in general the heaviest grades are in Oregon. The Table below† gives the dimensions and size of many of the different ditches in the various States and Territories where ditches are used for mining purposes.

COUNTY AND DISTRICT.	DITCH.	Grade.	Width at Top.	Width at Bottom.	Depth.
	ALABAMA.				
CLAY.		ft. p. m.	ft.	ft.	ft.
	Idaho	$13\frac{1}{3}$	2	$1\frac{1}{2}$	$1\frac{1}{2}$
	ARIZONA.				
APACHE.					
Greenlee Gold Mountain	Boston	13	5	$2\frac{1}{2}$	$2\frac{1}{2}$
	CALIFORNIA.				
BUTTE.					
Cherokee	Spring Valley	12	9	6	$3\frac{1}{2}$
Oroville... ...	... Feather River and Ophir	11	9	6	3

* Report of U.S. Census for 1880, vol. xiii., p. 222.

† *Ibid.*, pp. 223-225.

County and District.	Ditch.	Grade.	Width at Top.	Width at Bottom.	Depth.
	CALIFORNIA—*con.*				
Calaveras.					
Mokelumne Hill	Cook and Clarke ...	12	3	2½	1⅝
	Mokelumne and Campo Seco	12	9	5	2
Del Norte.					
	Bunker Hill	13⅓	5	4	2
	China Creek, No. 2 ...	18⅓	3½	2	1⅜
	,, ,, No. 3 ...	18⅓	2½	1⅜	1½
	,, ,, No. 4 ...	18⅓	6	4	2
	Happy Camp, No. 2 ...	13⅓	6	5	2½
	Muc-a-muc	16⅔	4½	2½	1⅝
	Wingate, No. 3	13⅓	5	4	2
El Dorado.					
Georgetown	California	6 to 16	3½ to 8	2 to 5	1½ to 3
	El Dorado	4	10	6	4
Placer.					
Gold Run	Gold Run	26	6	4	2½
Iowa Hill	Iowa Hill	10	10	7	4½
Plumas.					
Seneca	Dutch Hill	40	6	3	3
	Hungarian Hill	12	6	2½	2½
	Plumas	13⅓	7	5	3½
Siskiyou.					
Indian Creek	Bay City, No. 2... ...	13⅓	3	2	1½
,, ,, ...	Coyote Creek, No. 3 ...	13⅓	4½	3	2
Oro Fino...	Altoona	16⅔	6	4	3½
,, ...	Wright and Fletcher ...	13⅓	10	8	3
,, ...	Young's. No. 2	13⅓	10	7	2
Quartz Valley	Beaver Dam	6⅔	5	3½	3
,, ,, ...	Shackleford Creek ...	6⅔	7	5	4
Sciad Valley	Fort Goff, No. 1 ...	26⅔	4	3	2
,, ,, ...	,, ,, No. 2 ...	26⅔	6	4	2
Ureka	Pellett and Truitt, No. 2	6⅔	3	2	1
Trinity.					
Buckeye...	Buckeye...	10	5	3	3
Indian Creek	Johnston	10	5½	4½	3
Junction City	Lewiston...	...	5	3	3
Taylor's Flat	Trinity	10	7	5	3½
	,, G. M. Co. ...	11⅛	7	4	3
Tuolumne.					
Columbia	Tuolumne	11 to 32	11 to 15	7½ to 11	4
	COLORADO.				
Lake.					
California	Stevens and Leiter ...	5	10	7	3
Hahn's Peak	Elk River	10	4	3	2½

County and District.	Ditch.	Grade.	Width at Top.	Width at Bottom.	Depth.
	COLORADO—*con.*				
Summit.	Boston G. and S. M. Co.	7	2½	...	1⅔
	Gold Run	6⅔	2½	...	2
	Sisler	6 to 8	3	...	1¼
	Union French	6⅔	4	...	1⅔
	DAKOTA.				
Lawrence.					
Cape Horn	McKay and Ross ...	27½	3	...	1½
Confederate	Stockade...	3⅓			
Whitewood	Black Hills C. and W. Co.	13¾ to 27½			
Pennington.					
Cañon	Last Chance	6⅞			
	GEORGIA.				
Dawson.					
Fourth, Fifth, and Thirteenth...	Annicolola and Etowah (main)...	4	6	5	3
Lumpkin.					
Twelfth	Dahlonega	4	3	2	2½
,, ...	Hand, No. 1	4	6	5	3
,, ...	,, No. 2	4	5	4	3
,, ...	Singleton	4	5	3	1½
Third	Nacoochee	5	5	3	3
	IDAHO.				
Boise.					
Boston	Spencer	16⅔	4½	3	2½
Moore's Creek	Alderson	16⅔	4	3	2
,, ...	Dunn	13⅓	4	3	2
,, ...	Christy	10	5½	4	2
,, ...	Crepisculla (main) ...	6⅔ to 10	3 to 4	2 to 3	2
,, ...	Cuddy	16⅔	3½	2½	2
,, ...	Plowman	10 to 13⅓	5	3½	2
,, ...	Thorn Creek	10 to 13⅓	5	3½	3
Pioneer	Stevenson & Noble, No. 1	26⅔	6	4	3
,, ...	Wilson, No. 1	10 to 13⅓	8½	6	2
,, ...	,, (others)... ...	10 to 13⅓	5½	4	2
	MONTANA.				
Beaver Head.					
Bannock	Bannock D. and M. Co.	8	5	4	2½
,, ...	White Bar	6⅔	5	3	1⅔
Deer Lodge.					
Henderson Gulch ...	Smart's Creek	6⅔	7	5	2
Nelson	Pioneer and Keystone ...	13⅓ to 16⅔	4	...	2½
Squaw Gulch	Rock Creek	10	7	...	3½
Summit Valley	John Noyes	13½			
Lewis and Clarke.					
Last Chance	Wm A. Cheesman. No. 1	5	7	4	2

County and District.	Ditch.	Grade.	Width at Top.	Width at Bottom.	Depth.
	MONTANA—*con.*				
Lewis and Clarke.					
Last Chance	William A. Cheesman, No. 2.	5	8½	5	2½
Meagher.					
German	Diamond Flume and Hydraulic Co.	13⅓	7	5	3½
Thompson Gulch ...	Birch Creek	20	2½	1⅜	1⅝
,, ,, ...	Camas ,, ...	13⅓	3	1⅜	1½
	OREGON.				
Baker.					
Blue Cañon	Littlefield & Duckworth	13⅓	3	2	2
,, ...	Marysville	13⅓	5	3	2
Clarksville	Placer G. M. Co., No. 1	6⅔	6	4	2
,, ...	,, ,, No. 2	20	8	4½	2½
Pocahontas	Baisly	20	5	4	1½
,, ...	Jones and Carpenter (upper)	33⅓	2½	2	1½
,, ...	Jones and Carpenter (lower)	33⅓	5	3	2½
,, ...	Lew Cooper and Co. ...	13⅓	1⅔	1¼	1
,, ...	Nelson	20	6	4	3
Shasta	El Dorado (main) ...	8 to 12	7	5	2
,, ...	,, (others) ...	8 to 12	2 to 2½	1½	1 to 1½
Sumter	Young and Rimbol ...	9⅙	4½	4	2
Coos.					
	Pioneer	7⅓	4	3	3
Douglas.					
Big Bend Cow Creek ...	J. G. Fullers (new) ...	17¾	3	2	2
,, ,, ...	,, ,, (old) ...	...	2	1	1⅔
,, ,, ...	D. F. Salmon (main) ...	10	3½	2½	2
,, ,, ...	,, ,, (others)...	13⅓	2	1⅔	1⅔
Grant.					
Cañon City	Humboldt	20	8	3	2
,, ...	Wolfinger's	6⅔	6	4	2
Elk Creek	Deep Creek	13⅓	4	3½	1¼
Granite	Borne and Lucas (main)	40	4	3	2
,, ...	,, ,, (others)	40	2½	2	1½
,, ...	J. B. Gardner's (main)...	13⅓	5	4	1½
Marysville	Hillis	5⅓	12	6	7
,, ...	Miner's	5⅓	6	3	2½
,, ...	Prairie Diggings ...	26⅔	4	3	1⅝
,, ...	Thompson's	45⅞	4	3	1⅝
Rock Creek	Lake	10	5	3	3
,, ...	Rock Creek	27½	4	3	2
Jackson.					
Applegate	Grand Applegate ...	20	6	3	3
,, ...	Squaw Lake	13¼	5	3	2¼

County and District.	Ditch.	Grade.	Width at Top.	Width at Bottom.	Depth.
	OREGON—*con.*				
Jackson.					
Coyote Creek	William Rush	17¾	2½	1⅜	1⅜
Farris Gulch	Farris Gulch (upper) ...	13¼	4	2⅛	2
Forty-nine	Ashland	20	7	3	2
,, ...	Forty-nine (Coleman) ...	26⅔	5	1½	2
,, ...	,, (Anderson) ...	20	4	2	1¼
,, ...	,, (Wagner) ...	13¼	5	3	2
Sam's Valley	Hays and Magruder ...	6⅔	3	2	1⅔
Sterling	Sterling	12⅕	8	4	3
Uniontown	Chapel and Co.	13	3	2½	2
,, ...	Gin Lin	13⅓	6	3	2½
Wolf Creek	Gross Brothers	20	3	2	1½
Josephine.					
Grass Creek	Goff, Triplett & Chapin, No. 1	26⅔	3	2	1½
,, ...	Goff, Triplett & Chapin, No. 2	26⅔	1½	...	1⅓
,, ...	Steam Beer	16⅔	3½	2	2
Murphy	Carson and Johnson ...	13¼	3½	3	1
Waldo	Desselles, No. 1 ...	13¼	6	4	3
,, ...	,, No. 2 ...	13⅓	3½	3	2
,, ...	Waldo (main)	6⅔	7	5	3
,, ...	,, (pipe ditch) ...	6⅔	3½	2½	2
,, ...	,, (waste water ditch)	6⅔	7	5	2½
Yank	Blue Gravel	20	5	3	3

As a general rule, no lining is put into the ditches now constructed. The places where it appears likely that a lining will be required are avoided. It is only when a ditch has already been constructed, when defects appear in its working, that a lining is used. The El Dorado Ditch Company has two or three miles lined with granite and slate. The Park Canal is well puddled, but in places is lined with timber. The Ridgefield Ditch, for a mile, where it has a grade of 40 ft. to the mile, is walled with granite. The rest is cribbed with split rails for 10½ miles. The current varies from six to eight miles an hour. The Trinity Hydraulic Mining Company's ditch is lined with 1¼ in. lumber 15 in. wide. The Oro Mining Ditch and Flume Company, Lake County, Colorado, is lined with sod for four miles. The Black Hills Canal Company has underground box lines made of pine and covered with 3 ft. of earth to prevent freezing in the winter and evaporation in the summer. These are about all the methods in use. Their application depends upon circumstances, and it may be said that with care in

the original construction of the ditches most of these could have been avoided, except where for any reason the current is to be rapid, and where a lining must be put to prevent the wash of the banks.

In the whole United States only 11 out of the 128 companies who were consulted* reported that they lined their ditches in any part, the fine silt held in suspension by the water of the ditch forming in time, without artificial puddling, though this is sometimes resorted to, a clayey covering on the inner surfaces sufficiently impermeable for practical purposes. It is from this cause that, within reasonable limits and with proper repairs the carrying capacity increases considerably with the age of the ditch, the loss from leakage gradually diminishing. This result is also furthered by the settling of the earth of the outer embankment which, as it becomes more compact, is less liable to leakage and at the same time presents a better surface for the flow of water.

In following the contour of the hill-sides, to maintain a constant grade without resorting to the expedient of flumes, sharp turns are often unavoidable, as in crossing the head of ravines or in rounding a sharply projecting flank of the hill. It is at such points that eddies are formed and the greatest exposure occurs. If the embankment is here in danger of being washed out, it may be protected by cribbing with timber and logs, or by facing with stone or boards at the salients. After a new ditch has been in operation a short time these weak spots manifest themselves and are fortified as occasion demands, the whole line being under very careful scrutiny until its permanence is established. At re-entering angles, threatened with damage by small water channels crossing the line of the ditch, the indications given in advance of construction are usually plain enough to cause suitable precaution to be taken. They are often protected by throwing one or more boxes of fluming over the exposed point, and deepening the natural channel beneath, &c. When it is possible, these side waters are, of course, turned directly into the ditch, to add to its contents and off-set the loss from leakage or evaporation, care being taken to bring the confluents in at easy angles and to protect the outer bank from the impinging

* Report of the U.S. Census, 1880, p. 226.

cross currents. The experienced ditch builder utilises these advantages or avoids the dangers as if by instinct.

The repairs to the ditches are caused mostly by small breaks and washouts, either from freshets or improper filling of the ditches or too rapid a stream. In the well-established lines the accidents, owing to careful supervision, are very few in number. After a very few months well-built ditches have their sides and bottoms so well coated with fine sediment and their banks so thoroughly solidified by age that they lose very little by leakage, and are easily kept in repair unless they are of very large size and the ground very porous. Even then they require only watching to ascertain where and when the leakage begins, in order to stop it before disaster occurs. The most frequent causes are streams, land slides, and the boring either of crayfish or other amphibious or land animals. One of the most frequent sources of leakage are the holes formed by decayed roots, or the filtration of the water beside the roots left in the ground. It is therefore necessary that trees found on the line of the ditch should not be cut down, as they usually are, but they should be undermined, the small roots cut off, and the tree with all its roots thrown down the side of the hill. When the commencement of a leak is not quickly discovered, it may lead to a washout, and if the cause cannot be prevented, the ditch is repaired by putting up flumes on the part so affected. When land slides can be foreseen the locality is either walled, or, when that cannot be, willows or other trees are planted on the upper side to hold the bank. The damage from snow-slides is averted by placing a low shed, called a "snow-guard," built of very strong timbers, over that part of the ditch which has a steep hill above it. The snow at such points pushing down slowly into the ditch might fill it completely, or if it came suddenly might carry it away entirely. When only a light snow is to be feared, the ditch may be protected by covering the whole length of it with branches, so that the slide goes over rather than into the stream. As there is no possibility of telling when a greater amount of water than the ditch can carry will be turned into it, either from sudden thaws or floods, overflows, carefully built, with or without gates, should be put

in each ditch from time to time to let off the surplus water, and every effort made to ascertain the very beginnings of accidents, and to check them, so that what may cause a serious trouble, if left to continue, may be prevented. Notwithstanding their liability to them, the number of accidents which actually do happen is very small. While each accident is of very little consequence in itself, in very long lines of ditches they cause a very considerable amount of expenditure.

The cleaning of the canal of the El Dorado Ditch Company, which is forty miles long, and with a grade of 4 ft. to the mile, cost $800 per month. The repairs to the Park Canal cost about $165 per month. The repairs during the year on the Mokelumne and Campo Seco line, in Calaveras County, California,* for instance, amounted to $6000, as against a first cost of $50,000, or 12 per cent.; on the Round Valley, in Plumas County, they reached 6 per cent.; on the Plumas, in the same county, $10,000 on an investment of $50,600, or 20 per cent., including, however, a thorough regrading of the ditch; the Rock Creek line, in Deer Lodge County, Montana, $2500 in repairs, against $20,000 first cost, or 12½ per cent.; and many others ranged from 3 to 15 per cent. Among lines calling for large expenses of this kind may be mentioned the Spring Valley, $5000 per year; the Morris Ravine, $4500; the Happy Camp, $1000; the California, $2500; the El Dorado, $9600; the Park Canal, $2000; the South Yuba, $15,000; the Gold Run, $2000; the Iowa Hill, $3000; and the Tuolumne, $8000—all in California; the Elk River, in Colorado, $2400; the East Moore's Creek, $1000, and Plowman, $1400, in Idaho; the Bannock, $1000, in Montana; and the El Dorado, in Oregon, $6240; the Osborne, Placer Gold, Borne and Lucas, Squaw Lake, Farris Gulch, and Sterling lines entailing an annual expense for repairs of $1000 each or over. The smaller, cheaper, and less permanent lines require lighter annual expenditure per mile for repairs than the plant of the greater engineering enterprises, but these repairs amount to a higher percentage upon first cost. The yearly improvements, other than repairs included under the head of general labour, upon 2836½ miles of ditch, amounted to $114,365.50, or $40.32 per mile. This corresponds to a yearly tax

* Report of the U.S. Census, 1880, vol. xiii., p. 236.

of 1.06 per cent. upon an original investment of $10,780,414.31. A similar rate of expense is assumed to have been incurred by 213 additional miles, bringing the total for 3049½ miles up to $122,953.66. In the absence of data for estimating what proportion of the repairs incidental in operating 7733½ miles of ditch included in the total plant are to be assumed as additional to the cost of regular attendance, or even what proportion were in active use, the arbitrary ratio of $10 per mile is assumed as being within the probabilities. As the lines not included in the census examinations were, as a rule, of the less important class, it is not probable that their cost of maintenance would be so high as that of the scheduled lines. To recapitulate:

Annual cost of repairs on 2836½ miles of ditch ...	$114,365.50
Estimated annual cost of repairs on 213 miles but without specified cost	8,588.16
Estimated annual cost of repairs on 7733½ miles ...	77,335.00
Total	$200,288,66

The amount of water delivered by 173 lines as reported upon to the United States Census is 184,088,958 gallons per hour, corresponding to 262,984 miners' inches, and it is presumed that the figures reported in the majority of cases are of the maximum capacity, not the actual average delivery of the several lines. The average delivery per ditch of these lines in the height of the season is 1,064,098 gallons per hour, or 1520 miners' inches. For the whole number of ditch lines in operation in the United States, including ditches not reported upon, the maximum capacity is estimated at not less than 450,000 in., or 315,000,000 gallons per hour and 7,560,000,000 gallons per twenty-four hours. This is about eighty times the capacity of the old Croton Aqueduct, which supplies New York City with water. The amount of water actually delivered is, however, very variable, and the average during the season is far below the maximum capacity of the ditches.

The average amount of water lost in transit through the ditches is believed to be, by leakage, 7.40 per cent.,* and by evaporation, 9.06 per cent. of the quantity actually delivered. The statements selected by the United States Census in making this

* Report of the U.S. Census, 1880, vol. xiii., p. 220.

computation were those in which the losses by both leakage and evaporation were specifically stated, and many of the answers gave the total supposed loss, but did not discriminate as to the cause. While these results have not the weight of careful experiment, they represent the current opinion of well-informed persons. With well-built and established lines the loss by leakage is usually very small, though new lines, cut in light soil, are apt to leak badly for a time, until their walls have thoroughly settled and the bad spots have been strengthened. Evaporation depends upon the length of the line; that is, the time during which the water is exposed in transit; upon the location of the ditch, whether on the north or the south slopes of the hills, and whether shaded or partially shaded by timber; upon the climate and season, the latitude and the altitude, and upon the porosity of the ditch banks. The earth at the sides of the ditches acts as a sponge, absorbing water, and by capillary attraction bringing it under the influence of the sun's rays, thus increasing the exposed water area. It is therefore not remarkable that the loss by evaporation should as a general average exceed that by leakage, though in particular cases the circumstances are reversed. The variation according to season is naturally very marked, not only because of the difference in intensity of the solar heat, but also because of the difference in humidity of earth and air in the rainy and the dry portions of the year. The average total loss by both leakage and evaporation is 16.46 per cent., calculated upon the amount of water delivered.

The amount of water, actually delivered, during the summer season is, however, often reduced to nearly one-third of its normal quantity. It is therefore desirable, if possible, to avoid all sources of loss; but as this cannot be done, all the small streams on the line of the ditch should be made to flow into it as tributaries, but where it is liable to be crossed by torrents, it should be carried over the bed of these streams at a point sufficiently high to insure that the highest waters will not reach it. As the streams, with the dam supply, will be likely to furnish an excess of water during the rainy season, floodgates should be provided at fixed distances which may be opened to allow the excess of water to run off, so as not to

endanger the ditch. They can be closed when the water supply is deficient. This arrangement will allow of rapidly repairing the ditch when a break actually occurs, as the water can be easily turned off from any section needing repairs. Overflows should also be provided, so that a slight excess of water may not damage the banks. All these water outlets should be so protected that there will be no danger at any time of the water undermining the banks of the ditch.

It is usual in constructing the ditches to commence them in sections simultaneously over all parts of the line, but it is generally best to finish the upper end first, so that it can be used as a canal to float down the lumber required for the lower sections. Generally the best timber is found in the upper part of the line, and much time, expense, and labour is gained in cutting it there and floating it down to where it is to be used, instead of hauling it over such roads as are possible in such regions. In many places no other means of transporting the timber would be possible.

The Table on page 135* gives the total length of the ditch lines of California by counties according to the official returns of the State Surveyor-General, the figures for 1870, from the same source, being annexed for the purpose of comparison. The increase in the aggregate length and the decrease in the number of lines in the decade indicate a tendency towards consolidation of interests and also towards operating on a much larger scale than heretofore. In 1870 the average length per ditch was about $8\frac{1}{5}$ miles; in 1880 it had increased to $12\frac{1}{2}$ miles, the increase in the average being largely attributable to the construction of a few lines, each of which is greatly in excess of the present average. In this connection it may be remarked that the average length of the California ditch lines is slightly over twenty-six miles.

These ditches are capable of supplying from 400 to 500 up to 6000 to 7000 miners' inches of water. The main trunk of the South Yuba Canal, extending from the head dam on the South Fork of Yuba River to Bear River, a distance of one and a half miles, transmits for that distance 7000 in. of water. This canal

* Report of the U.S. Census, vol. xiii., p. 210.

is here 6 ft. wide on top, 5 ft. deep, and has a fall of 13 ft. to the mile. The Chalk Bluff branch of this ditch carries 4000 in. of water. The company owning it supplies during most of the year 200,000,000 gallons of water daily. The length of the single ditches is very variable, some of them are not more

County.	1870.			1880.	
	Number of Ditch Lines.	Length.	Amount of Water used per Day.	Number of Ditch Lines.	Length.
		miles.	in.		miles.
Amador	32	405	5,450	7	300
Butte	32	190	41,000	...	390
Calaveras	19	515	5,800	26	525
Del Norte	30	112	8,912	22	87
El Dorado	58	966	9,450	5	500
Inyo	1	3	40	1	6
Kern	2	5	1,728		
Klamath	80	90	16,000		
Lassen	...	...	...	3	7
Los Angeles	3	25	800	4	21
Mariposa	10	66	Unknown		107
Mono	2	13	720	5	40
Nevada	70	946	42,400	217	824
Placer	10	302	14,000	22	358
Plumas	...	...	...	34	700
Sacramento	2	45	4,000	2	200
San Bernardino	2	5	500		
Shasta	...	...	...	28	650
Sierra	49	208	21,000	55	366
Siskiyou	24	290	5,010	6	250
Stanislaus	3	12	1,000	1	15
Tehama	1	7	160		
Trinity	116	340	45,773	25	400
Tulare	...	...	...	1	
Tuolumne	...	...	...	14	190
Yuba	18	69	4,500	6	100
Total	564	4614	228,243	484	6036

than three to four miles long, a number are forty to fifty, and a few seventy miles in length.

The cost of ditch building varies very greatly, depending upon their size and length, the nature of the soil, and the engineering difficulties encountered. Generally the proportional cost of large ditches is greater than small ones. Including dams, flumes, and pipes, it may be as much as $15,000 per mile. For moderately small ones it may vary from $4000 to $5000. Occasionally they

cost as little as $2000 per mile. This may be owing to rock-cutting, tunnelling, &c. A porous soil must be puddled or otherwise filled. When the ground is full of springs, these must be drained. When the bedrock is very hard, inclined, or slippery, it must be cut out. It is almost impossible to foresee what these difficulties will be. The great engineering obstacles which can be foreseen can be studied beforehand and counted in the cost, but innumerable little things, which become serious difficulties at unexpected times and places, cannot be guarded against or provided for in the estimates. The entire cost of the ditches in actual use in California alone at the time of the injunction was probably not less than $30,000,000. This estimate does not include many of the costly structures which were at that time abandoned.

The lowest cost per yard reported to the United States Census* is Yreka, California, where the cost is reported as being only 5 cents. Cases of 10 to 20 are common in both California and Georgia. Cases of 20 to 50 cents are common in California, Georgia, Idaho, and Oregon. Every one of the eleven States and Territories using ditches report many cases under $1, which is far above the average. Six cases only are reported between $1 and $2, only four between $2 and $3, and two between $3 and $4. One at Waldo, Josephine County, Oregon, cost $4; one at Columbia, Tuolumne County, cost $4.50; one at Georgetown, El Dorado County, California, $5 to $5.50; and one at Sherman, Meagher County, Montana, $6.50. The average cost is below $1. Most of the expensive ditches are in California, owing to the fact that they have there had some of the greatest difficulties to contend with, and that some of their engineers have been looking rather to *tour de force* in engineering construction, than to economy. These ditches have not always been successful, though generally when they have failed, it has been on account of the same want of foresight which would have wrecked almost any business enterprise. A notable instance of this was the Truckie Ditch, capable of carrying 3000 in. of water. It was a very costly enterprise, having over thirteen miles of flume, some of it suspended from the sides of

* Report of U.S. Census, 1880, vol. xiii., p. 212.

a cañon. It cost about $1,000,000, but when it was finished it was found that there was so little demand for the water, a fact which ought to have been determined beforehand, that it was abandoned. Such failures, though not very frequent, occurred in the earlier days, but are very uncommon now, when these enterprises demand great business ability, as well as great engineering skill, which they always have had.

The following Table* gives interesting details of some of the most important ditches in California:

Name.	Length.	Capacity.	Grade, per Mile.	Cost.	Dimensions.		
					Top.	Bottom.	Depth.
	miles	in.	ft.	$	ft.	ft.	ft.
North Bloomfield, including reservoirs	157	3,200	12 to 16	708,841	8¾	5	3½
Milton, including reservoirs	80	3,000	12 to 25	391,579	6	4	3½
Eureka Lake and Yuba	163	5,800	...	723,342			
Smartsville Ditches	...	5,000	9	1,000,000	8	5	4
Hendricks	46½	...	6 to 12	136,150	5	...	2
La Grange†	20	2,700	7 to 8	500,000	9	6	4
Blue Tent	32	1,800	10	150,000	8	6	4
Total	498½	21,500	6 to 25	3,609,912	5 to 9	4 to 6	2 to 4

Statement of *Construction of Milton Ditch, Eureka to Milton, built by the North Bloomfield Gravel Mining Company in the Years* 1872, 1873, 1874.‡

Length.	Chains.	Miles.
Eureka to South Fork	563	7.04
South Fork to Drop-off	96	1.20
Drop-off to Milton Dam	884	11.17
Total distance	1,543	19.41 or 102,498 ft.

Fluming.	Twelve-foot Boxes.	Linear Feet.
Eureka to South Fork	961	11,536
South Fork to Big Bluffs	264	3,168
Big Bluffs to Milton	1113	13,352
Total	2338	28,056

* Report of U.S. Census, 1880, vol. xiii., p. 209.

† Most of this ditch is hewn out of granite.

‡ Mining Commissioners' Report, 1875, p. 123.

The above 2338 boxes include fifty-six boxes built in the ditch, and most of which are supported by heavy cribbing. In addition to the above, there are several small branch flumes, one large crossing flume, and 130 ft. of ditch lining.

A dam was built at South Fork at a cost of, say, $350; old dam at Milton was used.

Waste-ways.			ft.
Eureka to South Fork	14	wastes, aggregating	112
South Fork to Big Bluffs	12	,, ,,	48
Big Bluffs to Milton	24	,, ,,	114
Total	50	,, ,,	274

COST OF MILTON DITCH, FROM EUREKA TO MILTON, 19.41 MILES.

Excavation, &c.							—
	Distance.	Labour.	Explosives.	Tools.	Steel.	Coal.	
	miles.						
Ditch .	14.1	$69,664.92	$4098.46	$1606.67	$319.48	$953.38	$76,642.91
Flume foundation	5.3	15,013.40	2866.72	525.50	213.00	301.11	18,919.73
Clearing line .	19.4	3,582.01	...	90.00	...	...	3,672.01
	19.4	$88,260.33	$6965.18	$2222.17	$532.48	$1254.49	$99,234.65

Flume.	ft.	ft.		
Lumber, &c., Milton to Lower End Big Bluff	1,083,434			
Less sold to Milton Company	200,000			
		883,434		
Eureka to Big Bluffs, 11,225 boxes		765,911		
Total on hand and used for 2338 boxes ...		1,649,345	$32,015.28	
NOTE.—Of the above amount of 883,434 ft. it is supposed that there is on hand, say, 130,000 ft., thus leaving 750,000 ft. as the amount used for 1113 boxes from Milton to lower end of Bluffs.				
imbers cut by hand, stringers, posts, &c.			1,301.49	
Hauling lumber to Milton, Little Poor Man's, &c.			1,650.00	
				$34,966.77

Carpenters, &c.

Gang.	Boxes 12 ft. long.	Labour.	Nails and Iron.	Tools.	Totals.	
Young . .	1145	$10,902.81	$1499.57	$50.00	$12,452.38	
Marriott. .	1193	10,497.90	1559.57	50.00	12,107.47	
	2338	$21,400.71	$3059.14	$100.00		$24,559.85
						$59,526.62

General Cost.		
Surveys.—Engineer, who was also foreman, and assistants		$4,610.50
Roads.—South Fork to Bowman's, 3¾ miles	$1200.00	
,, Little Poor Man's, 2½ miles	200.00	
		$1,400.00
Hauling.—Transportation of tools, material, and men		1,450.94
Boarding.—Loss in boarding labourers, who were charged 75 cents per day		685.75
General Expense.—Being a portion of North Bloomfield Gravel Mining Company's cost of management, office, taxes, &c., while ditch was being built		3,564.63
Recapitulation.		$11,711.82
Cost of excavation		99,234.65
Lumber		34,966.77
Carpenters and supplies		24,559.62
General expenses		11,711.82
Damages.		
Eureka Lake Company.—Damage to it by breaking its miners' ditch by blasts		$1,635.87
Total cost		$172,108.96
Collected from Milton Company for account extra work		689.30
Leaving Milton Ditch account, Nov. 10, 1875, on Company's books ...		$171,416.96

NOTE.—If the 130,000 ft. of lumber supposed to be at Milton is sold for cost, $20 per 1000 ft., the total cost of the ditch will be reduced to $169,508.96, or, say, $8700 per mile. In that event:

Cost per foot, &c.

Ditch, 74,442 ft. long, cost for, say, 117,600 cubic yards, $76,642.91 or 65 cents per cubic yard, or $1.03 per linear foot.

Flume, 28,056 ft. long { Cost for excavation, $18,219.73 or 67 cents per linear foot. / Cost for lumber, labour, &c., $59,526.62 or $2.12 per lin. ft. } $2.79 per linear foot.

The total length of mining ditches in the United States, in the year 1880,* was not less than 10,783 miles. This estimate includes both the large ditches of California, as well as the small ones of the East. Many of these ditches have involved engineering structures which have demanded not only ability but great engineering skill. The South Yuba Ditch is carried across Deer Creek by a wire suspension bridge, and across Squirrel Creek by a truss flume with a span of 60 ft. Such structures as these are not now usually built. It is generally easier to carry the water over in pipes. The thirty principal ditches of California have an aggregate length of 1600 miles, and carry 100,000 miners' inches of water, and cost about $20,000,000.

* Report of the U.S. Census, 1880, vol. xiii., p. 210.

SIZE AND COST OF DITCHES.

Name of Ditch.	Source of Supply.	Length in Miles.	Size of Ditch.			Capacity in Miners' Inches in Twenty-four Hours.	Grade to the Mile in Feet.	Cost.
			Top.	Bottom.	Depth.			
La Grange Ditch, Stanislaus Co.	Tuolumne River, 18 miles from the mine	25	9	6	4	2700	7 to 8	$450,000
Tuolumne Water Co.	South Fork of Stanislaus River	125*	11 to 15	7½ to 11	4	...	11 to 32	300,000
Union Water Co.	North " " "	40	...	...	...	2500	3 to 25	20,000
Mokelumne Hill Co.	Mokelumne River	100	...	...	...	1500	8 to 16	
El Dorado Gravel Co.	Silver Lake Amador Co.	110†	10	6	4	4000	4	600,000
California Water Co.	Loon Lake and Pilot Creek	250‡	3½ to 8	2 to 5	1½ to 3	1200	6 to 16	600,000
Park Canal Co.	Consumnes River	290§	8	5	2½	2200	½ to 16	2,000,000
Iowa Ditch Co.	North Fork American River	25	...	...	...	3000	...	200,000
North Bloomfield Co.	Big Can Creek and Middle Yuba River	157‖	8⅔	5	3½	3200	12 to 16	708,841
Milton Co.		80	6	4	3½	3000	12 to 25	391,579
Eureka Lake and Yuba Ditch Co.		163	...	...	...	5800	...	723,342
South Yuba Ditch Co.		123	8	4	4	7000	10	1,100,000
Smartsville Ditches.	... ...	...	8	5	4	5000	9	1,000,000
Spring Valley and Cherokee	... ...	52	...	...	...	2000		
Hendricks	... ...	46½	...	...	...	...	6 to 12	136,150
Blue Tent	... ...	31	...	...	...	2100	...	200,000

* Besides six miles of flumes. Ditches cost $4.50, flumes $14, and pipes $6.00 per yard.

† One mile of flumes, three miles of pipes; grade of flumes 1⅓ to 2½ ft. per mile. Reservoirs cost $45,000.

‡ Two and a half miles of flumes.

§ Flumes eight miles, pipe one mile. Ditch cost $10 per rod, flumes $12.14 per rod, pipes $2.50 to $4.50 per yard.

‖ It costs $13.463 a year to keep the reservoirs and ditches in order.

THE CHARACTER AND COST* OF MINING DITCHES TRIBUTARY TO THE SACRAMENTO AND SAN JOAQUIN VALLEYS.

Name.	County.	Character.	Length.	Capacity.	Cost.
			miles.	miners' inch.	$
Auburn and Bear River Canal	Placer	Mining and Irrigating	75	3,000	350,000
Amador Canal Company	Amador	,, ,,	66	2,000	400,000
Brandy City	Sierra	,, ,,	17	22,000	90,000
Cedar Creek, Dutch Flat	Placer	,, ,,	50	45,000	125,000
Cherokee and Spring Valley	Butte	,, ,,	56	22,000	500,000
California Water Company	El Dorado	,, ,,	125	4,500	550,000
Dardanelles Company	Placer	Mining	17	3,000	125,000
Excelsior, Smartsville	Yuba	Mining and Irrigating	110	5,300	1,000,000
El Dorado Canal	El Dorado	Mining	26	5,000	650,000
Gold Run Ditch	Placer	,,	26	2,500	150,000
Iowa Hill Company	,,	,,	27	4,500	175,000
Little York and Liberty Hill	,,	,,	35	3,500	150,000
Mokelumne Hill and Campo Seco	Calaveras	Mining and Irrigating	55	1,200	500,000
Natoma Canal	Sacramento	,, ,,	16	3,500	160,000
Phœnix Ditch Company	,,	,, ,,	100	4,000	880,000
Power's Ditch	Butte	,, ,,	30	2,000	75,000
Tuolumne Water Company	Tuolumne	,, ,,	75	3,600	1,500,000
Union Water Company	Sierra	,, ,,	50	1,500	350,000

* Prod. Precious Metals in the United States, 1883, p. 709.

The following Table* gives the quantity of water delivered by some of the type ditches of the United States:

QUANTITY OF WATER DELIVERED BY SOME MINING DITCHES.

County and District.	Line.	Quantity of Water delivered per Hour.
	ARIZONA.	gallons.
Apache.		
Greenlee Gold Mountain ...	Boston	300,000
	CALIFORNIA.	
Butte.		
Oroville	Feather River and Ophir ...	1,750,000
Calaveras.		
Mokelumne Hill	Cook and Clarke	1,260,000
	Mokelumne and Campo Seco	700,000
	Union	1,750,000
Del Norte.		
	Del Norte	1,296,000
	Happy Camp	3,283,000
El Dorado.		
Georgetown	California	8,400,000
	El Dorado	2,800,000
	Park Canal	1,540,000
Placer.		
Gold Run	Gold Run	1,505,000
Siskiyou.		
Indian Creek...	Bay City	594,000
,, ,, ...	Coyote Creek...	1,186,929
Oro Fino	Altoona	630,000
,, ...	Barker	490,000
,, ...	Wright and Fletcher ...	3,150,000
,, ...	Young's	3,110,000
Quartz Valley	Beaver Dam	560,000
,, ,, ...	Shackleford Creek	1,400,000
Sciad ,, ...	Thompson Creek	1,069,000
Trinity.		
	Trinity G. M. Co.	1,400,000
Tuolumne.		
Columbia	Tuolumne	2,100,000
Lake.	COLORADO.	
California	Star	275,000
,, ...	Stevens and Leiter	2,800,000
Park.		
Alma township	Alma Placer	275,000
Routt.		
Hahn's Peak	Elk River	1,400,000

* Report of the U.S. Census, 1880, p. 220.

County and District.	Line.	Quantity of Water delivered per Hour.
Summit.	COLORADO—*con.*	gallons.
	Boston G. and S. M. Co. ...	700,000
	Gold Run	700,000
	Sisler	196,000
	Union French	210,000
	DAKOTA.	
Custer.		
Battle Creek	Battle Creek Gravel M. Co.	472,500
Lawrence.		
Cape Horn	McKay and Ross	210,000
Confederate	Stockade	280,000
Whitewood	Black Hills C. and W. Co. ...	126,700
Pennington.		
Cañon	Last Chance	25,600
Rockerville	Black Hills Placer Co. ...	1,400,000
	GEORGIA.	
Hall.		
Twelfth	Glade	560,000
Lumpkin.		
Twelfth	Dahlonega	198,447
,, ...	Hand	1,023,694
,, ...	Singleton	198,447
White.		
Fourth	Town Creek	359,214
	IDAHO.	
Boise.		
Boston	Spencer	280,000
Moore's Creek	Alderson	700,000
,, ,, ...	Christy	560,000
,, ,, ...	Crepisculla (feeders)... ...	630,000
,, ,, ...	,, (main)	560,000
,, ,, ...	Cuddy...	280,000
,, ,, ...	Dunn	280,000
,, ,, ...	East Moore's Creek	700,000
,, ,, ...	Hanson	700,000
,, ,, ...	Lambing	700,000
,, ,, ...	Mountain	210,000
,, ,, ...	Plowman	490,000
,, ,, ...	Thorn Creek	700,000
,, ,, ...	Upper Alderson	210,000
Pioneer	Stevenson and Noble ...	2,800,000
,, ...	Wilson...	3,150,000
	MONTANA.	
Beaver Head.		
Bannock	Bannock	9,100,000
,, ...	White Bar	280,000

County and District.	Line.	Quantity of Water delivered per Hour.
Deer Lodge.	MONTANA—*con.*	gallons.
Henderson Gulch	Smart's Creek	560,000
Independence	Hail Columbia	175,000
Squaw Gulch...	Rock Creek	1,050,000
Summit Valley	John Noyes	
Lewis and Clarke		490,000
Last Chance	Wm. A. Cheesman	1,050,000
Meagher.		
German	Diamond Flume and Hydraulic Co.	1,750,000
Thompson's Gulch	Birch Creek	175,000
Baker.	OREGON.	
Blue Cañon	Littlefield and Duckworth ...	280,000
,, ,, ...	Marysville	700,000
Chicken Creek	J. N. Osborne and Co. ...	210,000
Clarksville	Placer Gold	1,120,000
Humboldt	Colt	210,000
Pocahontas	Baisley	420,000
,, ...	Jones and Carpenter ...	420,000
,, ...	Lew, Cooper, and Co. ...	175,000
,, ...	Nelson	490,000
Rye Valley	Powers	770,000
Shasta...	El Dorado	826,000
Sumter	Young and Rimbol	700,000
Willow Creek	W. Boswell	140,000
Douglas.		
Big Bend Cow Creek ...	J. G. Fuller	1,050,000
,, ,, ...	D. A. Leavans	834,900
,, ,, ...	D. F. Salmon	1,176,000
Cañonville	Catchings Brothers	371,000
Green Mountain	Charles Dwelley	9,000,000
Grant.		
Cañon City	Humboldt	700,000
Granite	Borne and Lucas	420,000
,, ...	J. B. Gardner	980,000
,, ...	Klopp and Johnson	1,050,000
Marysville	Hillis	560,000
,, ...	Miner's	210,000
,, ...	Prairie Diggings	350,000
,, ...	Thompson	525,000
Rock Creek	Lake	175,000
Jackson.		
Applegate	Squaw Lake	700,000
Sam's Valley	Hays and Magruder... ...	324,000
Sterling	Sterling	1,260,000
Uniontown	Gin Lin	1,400,000
Josephine.		
Waldo...	Desselles and Co.	2,494,000
,, ...	Waldo...	1,458,000
Yank	Blue Gravel	1,620,000
Wasco.		
Ochoco...	Wickaiser and Co.	140 000

The price paid for labour* varies greatly in different localities. In Georgia, as low as 50 cents per day is sometimes paid, while in Dakota, Idaho, and Montana, where the cost of living is necessarily very high, wages often reach $4. California, which contains the largest artificial water system, shows a medium status in respect to wages, the rates paid being sufficient to maintain the employés comfortably and yet not severely taxing the ownership. Wages at $3 per day in the latter State admit of better living than would $4 in the Territories. Chinese labourers receive much less than do the whites, usually from $1.30 to $2 per day; though in some cases as much as $3 per day is earned by skilled Chinese in Idaho. As a rule their pay averages from one-half to two-thirds that of white men engaged in the same character of work. It is customary in some districts for the employés to mess together, receiving board in part payment. The following statements give the range of salaries and wages, without board, in different parts of the country.

ARIZONA.—Ditch tenders, $60 per month to $3 per day.

CALIFORNIA.—Superintendents, $100 to $250 per month; collectors, $75 per month; bookkeepers, $50 per month; ditch builders, $3 per day; ditch tenders, $1.50 to $3 per day; Chinese labourers, $1.30 per day.

COLORADO.—Ditch tenders, $2.50 to $3 per day.

DAKOTA.—Ditch tenders, $2.50 to $4 per day.

GEORGIA.—Ditch tenders, 50 cents to $1.50 per day.

IDAHO.—Ditch builders and tenders, $4 per day; Chinese ditch cleaners and labourers, $2.25 to $3 per day.

MONTANA.—Ditch tenders, $60 per month to $4 per day.

OREGON.—Ditch tenders, $40 per month to $3 per day.

The length of a shift for ditch employés is from ten to twelve hours per day, more commonly the longer time, depending upon the number of hours run in the hydraulic mines supplied by the ditches. The average price per day for all employés, Chinese included, is $2.75 without board.

An examination of the reports from scattered districts shows an average of one person employed for every ten miles of ditch; but this does not include the irregular attendance upon ditches by men engaged for the greater part of their time in hydraulic mining, and who are consequently classed with the regular mine force. The wages paid to the force employed for a short period

* Report of U.S. Census, 1880, vol. xiii., p. 237.

at the beginning of the supply season in making repairs and overhauling the lines preparatory to active operations are also considered in the foregoing statement of cost of repairs. In the same way the men at work in constructing new lines are also excluded from the regular force of the ditches, as their wages appear under the heading of cost of plant. The wages of employés usually continue only during the water season, though a few of the more important lines, which do not furnish water during the whole year, keep a small force constantly occupied. The amount paid for attendance during the season of 1880 is estimated at $626,991.75, being the wages of 1078 employés of all classes for 211½ days at an average pay of $2.75 per day.

The labourers of the California placer mines are more than half Chinese, who, with their patient, persistent methods and practical business arrangements, usually gain fair wages. Although prevented by statute from mining on any public domain, and excluded from many districts by local laws, they manage, as they do on other claims, to either circumvent the law, or else to overcome obstacles, so that they gain a fair living. They do nothing for the country, trading only with their own people, using their own products, and sending all their gold out of the country.

V. The Construction of Flumes.

Where the ravines which the ditch must cross are not too high, or where a long stretch of level ground must be traversed in order to deliver the water to a certain height, and maintain the grade in places subject to constant breakage of the bank, owing either to depredation of animals or the natural weakness of the soil, or where the construction of a ditch would make the line too long, or where the sides of the ravine through which the water must be carried are either too steep or have too little earth to hold a ditch, or where the line must otherwise be cut through rock in order to avoid expense, the water is carried in launders or flumes, which may be smaller than the ditch and of higher grade than the ditch itself, as the smoothness of the boards produces less resistance or friction than a rough ditch, and the water consequently flows faster, or if they are of the same size, of a

little less grade, as there is less friction. The building of flumes or launders should be avoided wherever it is practicable, since they are not only likely to be destroyed by rot and by fire, but also to freeze in winter. They require much greater repair from the settling of the ground and the more or less rotting of the wood, which destroys the grade. Their average life does not exceed eight to ten years. Independently of the danger of rot and fire, they are, in high altitudes, likely to be crushed by the weight of the snow settling on them, or to be swept away by avalanches, which are only too common. There is no doubt that a well-built ditch, properly cared for, or a well-constructed pipe line, is cheaper in the long run than any flume that can be built, but the question with individuals and with small companies is not so much what the expense several years hence will be as what it will cost to-day. The money of to-day must come out of his or their pockets, while that of the future they hope can be earned from the ground, and so the makeshift is adopted as a matter of economy. Modern practice is, wherever it can be done, to use iron pipes, and, when considerable elevations must be crossed, to use iron pipe syphons, but there are some cases, especially where capital is small, where flumes must be built. Wherever flumes are to be built, all underbush, fallen trees, and every other combustible material, must be removed for a considerable distance on each side of their line, so as to protect them from fires in the dry seasons. Notwithstanding the fact that this precaution is generally taken, flumes frequently burn down.

They were formerly built not only a very great length but also a very great height, as shown in Fig. 14, sometimes reaching 100 ft. or 130 ft. One of them near Big Oak Flat, in Tuolumne County, was 1300 ft. long. Some of this flume was supported on wooden towers 256 ft. high.* On the Truckee Ditch there was at one time thirteen miles of flume, 8 ft. by 4 ft., hung to the sides of a deep cañon. On the Pilot Creek Ditch there was once a piece of flume 300 ft. long and 95 ft. high. The Myocene Ditch Company, of Butte County, in order to avoid the construction of a trestle, 180 ft. high, built a wooden flume, 4 ft. by 3 ft., which was carried around a bluff, 300 ft. high, suspended

* Min. Com. Rept., 1870, p. 476.

on iron brackets made of T rails fastened into holes drilled into the vertical face of the rock.

There are two general types of these flumes, which are known as trestle and bracket flumes; those which are built on trestle work have two varieties; those which carry a more or less heavy flume on trestle work more or less high, and the second where a mere launder is supported in some way as a more or less temporary structure. These flumes are of every variety from those supported principally on earth or rock to those hung almost in mid-air from vertical precipices. A flume which is not close under a rock or bank which can shelter it, must be protected in some way from slides. The sleepers on which the trestles rest should be made of well-seasoned wood, and should be either painted or tarred or charred. The trestles should be mortised into them. They should be from 8 ft. to 12 ft. or 16 ft. apart, but the closer together they are the better. The trestles are made of two legs, supported when they are quite high by one or more sets of stringers, or where they are not so large by simple cross-pieces. They are ordinarily made of pine, spruce, or fir, their size being proportioned to the duty they have to perform.

Fig. 14 shows a heavy structure near Smartsville, which is of much more costly construction than those usually built. Lighter structures of a much cheaper form are seen at almost every step in and about the gold and silver mines of California and Nevada. They are always objectionable and should be avoided, on account of the danger of their freezing in winter, and also because they are a constant source of expense. When the flumes are well protected and so close to the bank that there is no circulation of air, water will run, if the current is sufficiently high, without freezing even when the thermometer is far below zero Fahr., but when they were in the open, I have known them freeze solid at a much higher temperature. As they are generally much smaller than the ditch, and are to carry all its water, they should be set on a grade of 30 ft. to 35 ft. to the mile, in as straight lines as possible. Where curves are necessary they should be of very large radius. Launders and flumes are made of 1½ in. to 2 in. plank, 12 in. to 18 in. wide, with a framing

of 4 ft. by 4 ft. and 3 ft. by 4 ft. scantling for every 2½ ft. or 3 ft. The seams of the joints are covered on the outside by strips 3 in. wide and 1½ in. thick. They are supported every 4 ft. by a frame made of 4 in. by 4 in. timber; the sills in the heavy flumes extend 18 in., and are braced to the side; the lighter ones are braced to the uprights.

The strength of the supporting scaffolding, or bents upon which the flumes rest, must be proportional to their height and the

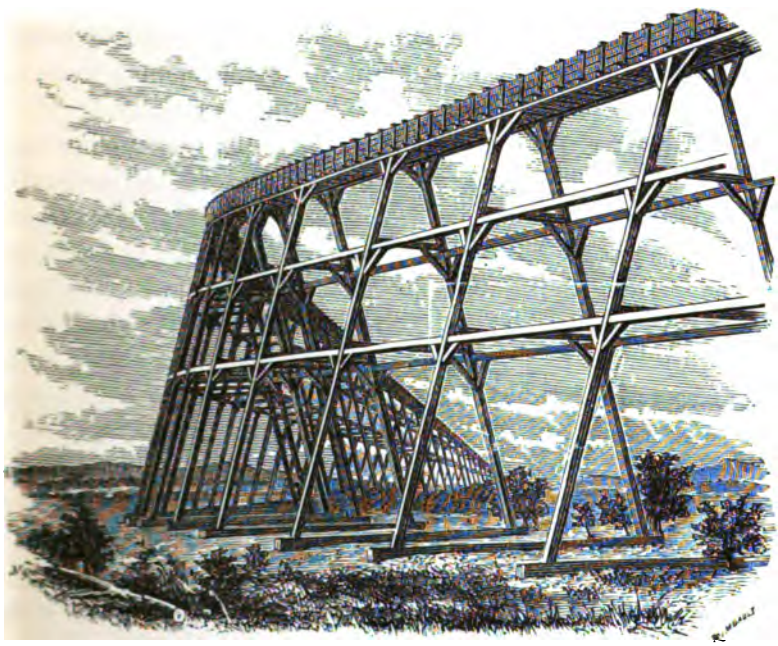


FIG. 14. FLUME NEAR SMARTSVILLE.

weight to be supported—both of the flume and the volume of water to be carried.

The greatest care should be taken with the foundations, and in placing the uprights, to see that the posts do not settle; they should be placed above the ground so that they should not rot. Nothing is so prejudicial to the proper supply of water as a change of grade in the flumes resulting from settling. Where the grade changes, there the water is likely to freeze, and I have known launders in Lake Superior, owing to settling from careless construction, freeze solid in the winter and necessitate their

being cut out for thousands of feet. Launders especially, as they are open to the effects of the winds upon all sides, are liable to freeze solid.

In order to protect them against swaying with the wind, when they are very high, they should be anchored with wire rope on both sides of the flume, secured at an angle of 45 deg. In all cases they should be in an open country if that is possible, as a protection against forest fires; or where this cannot be done, as where there is danger from snow slides, they should be set as close to the bank as may be, so that the snow may slide over and not against them. Brushwood and other inflammable material should be removed as far as possible from their vicinity.

The box of the flume must be made of the best lumber that can be had, sugar pine when it can be procured being preferred. The boards should be 1½ in. thick. The box varies in size according to the grade and the quantity of water it is to carry. It varies from 2 ft. by 1 ft. in small launders up to 8 ft. by 6 ft. in very large sized flumes. Generally all the flumes are wider than they are deep. Occasionally they are made nearly square. It rests in frames which are about 4 ft. apart. The thickness of the sides and bottom varies from 1 in. to 2 in. The planks are generally about 16 in. long. When the flumes are small, not over 20 in. deep, the sides are usually made of a single board, but they are generally larger than this, so that it is hardly possible to avoid the use of several boards. The seams, except in the corner bottom joint, are covered on the inside with battens 3 in. or 4 in. wide. Knot holes and splits and cracks are covered with short pieces in the same way. It is not often that the sides can be made of a single board only. Flumes made even of roughly split and badly seasoned lumber, after a little use, if kept constantly wet, run very nearly tight. The loss is occasioned by their being allowed to dry, when cracks form from shrinkage and the seams open. As sand is very apt to enter during this time, when they are wet again from the passage of the water, interstices which cannot swell tight on account of the presence of sand will be formed and the leakage will be likely to be great. It is consequently very bad practice ever to allow a flume to get dry. When difficult places have to be crossed, which would in-

volve either a very high trestle work or else an excavation, these flumes are sometimes suspended upon brackets. In order to avoid the necessity of such a trestle work 186 ft. high, which would have had to be built in crossing a deep cañon with vertical side walls, the flume of the Miocene Ditch Company, in Butte County, California, was carried through the cañon to its head. The wall of the cañon is vertical, and is 360 ft. high. For a distance of 486 ft. the flume was supported on iron brackets, Fig. 15, at a height of 100 ft. to 120 ft. above the bed of the ravine, and more than 200 ft. below the crest

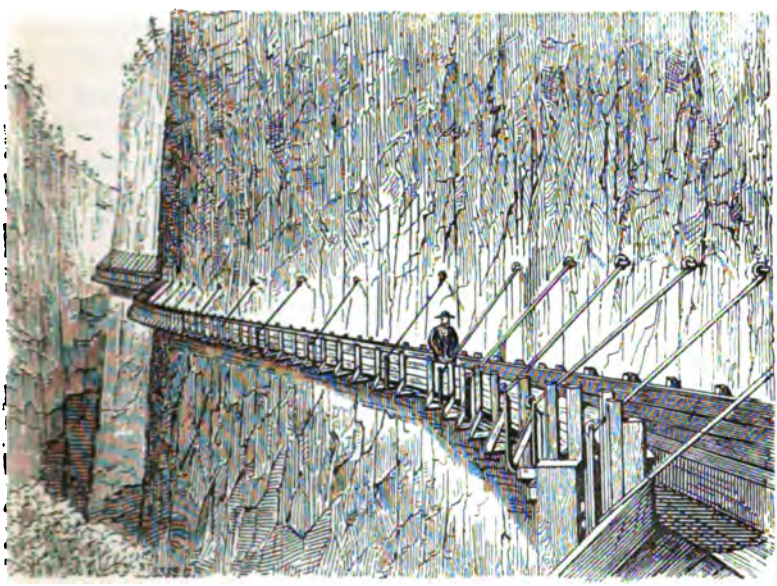


FIG. 15. FLUME OF THE MIOCENE DITCH COMPANY.

of the precipice. The brackets are made of 30-lb. T railroad rails bent up 2 ft. at one end into the form of an L, which was rounded off to let in ring-bolts; the other end was carefully clamped and fastened into holes drilled in the rock. The holes, as well as those for the suspended rods, were drilled by men swung from the top of the cliff by ropes and let down about 200 ft. The brackets are 8 ft. apart, and are capable of sustaining a weight of 14½ tons. The bed of the flume rests upon the brackets, the outer ends of which are turned up vertically. Rods of ¾-in

round iron are fastened to the ends of the uprights and bent diagonally upward and securely clamped and fastened to the rock. The flume is 4 ft. wide inside and 3 ft. deep, with a capacity of 3000 miners' inches. On this same ditch are two other similar flumes, one of which is 136 ft. high and the other 80 ft. high and 1088 ft. long.

Where the climate is severe and the water in the district is liable to freeze in the winter time, underground flumes or covered board ditches are resorted to. Great inconvenience resulted in the early days from the launders and flumes freezing solid, and in order to prevent this they are now sunk in the ground. They are simply a boxed ditch covered with earth. They do not require such careful construction as ordinary flumes, as they are covered on all sides by earth, which not only strengthens them but also diminishes the loss which can occur from leakage. When lumber is scarce the ditch is simply run in the earth, the top protected with branches or with boards, and covered with earth, which not only prevents freezing in winter but evaporation in summer. The objections to this system, on account of the difficulty of inspecting and cleaning the flumes, have prevented the use of this construction except where necessity compels it. It is generally desirable to have 2 ft. or 3 ft. of earth over the top of such flumes in order to prevent the penetration of the frost. Short lengths of such flumes as these are very common where it is necessary to pass under roads or buildings.

The grade of flumes is usually greater than that of the ditches of which they form a part, so as to make the flumes correspondingly smaller. The increase in the current gives a greater supply, so that more water can be passed through a small flume where the current is rapid than through a larger ditch with a less grade. The usual grades are 25 ft. to 30 ft. Occasionally, however, the higher grades are considered expedient for special purposes. The velocity of the water varies from three to six miles an hour. Occasionally it is as high as seven and a half, but this is very uncommon. The Table* annexed gives the grade and dimensions of some of the most prominent flumes in the United States.

* Report of U.S. Census, 1880, vol. xiii., p. 230.

GRADE AND DIMENSIONS OF FLUMES.

County and District.	Line.	Grade.	Width.	Depth.
	ALABAMA.			
Clay.		ft. p. m.	ft.	ft.
	Idaho	13⅓	2	1
	ARIZONA.			
Apache.				
Greenlee Gold Mountain	Boston	13	5	2½
	CALIFORNIA.			
Butte.				
Cherokee	Spring Valley	12	5	3½
Morris Ravine	Morris Ravine	9	5	2
Oroville	Feather River and Ophir	11	6	3
Calaveras.				
Mokelumne Hill ...	Cook and Clarke ...	12	3	1⅝
Del Norte.				
	Bunker Hill	13⅓	4	2
	China Creek, No. 1 ...	18⅓	2½	1¼
	,, ,, No. 2 ...	18⅓	3	1½
	,, ,, No. 3 ...	18⅓	2	1¼
	,, ,, No. 4 ...	18⅓	4	2
	,, ,, No. 5 ...	18⅓	5	2
	Muc-a-muc	13⅓	3⅜	1⅞
	Wingate, No. 1... ...	13⅓	2½	2
	,, No. 2... ...	13⅓	3	3
	,, No. 3... ...	13⅓	3	3
El Dorado.				
Georgetown	California	8 to 16	1½ to 4½	1½ to 4½
	El Dorado		8	6
	Park Canal		6	2½
Placer.				
Gold Run	Gold Run	12	4	3
Iowa Hill	Iowa Hill	10	7	4½
Plumas.				
Indian Valley	Round Valley	10	3⅛	1½
Seneca	Dutch Hill	40	6	4
	Plumas	26⅔	4	3
Siskiyou.				
Indian Creek	Bay City, No. 1 ...	13⅓	1½	1½
,, ...	,, ,, No. 2 ...	13⅓	2	1½
,, ...	Coyote Creek, No. 1 ...	13⅓	1½	1½
,, ...	,, ,, No. 2 ...	13⅓	1⅞	1⅝
,, ...	,, ,, No. 3 ...	13⅓	1⅝	1½
McAdam's Creek ...	Hiyou Gulch	13⅓	3	2
Oro Fino	Altoona	16⅔	5	2
,, ...	Barker	6⅔	4	2
,, ...	Wright and Fletcher ...	13⅓	7	2⅔
Quartz Valley	Shalkelford Creek ...	13⅓	6	2

County and District.	Line.	Grade.	Width.	Depth.
	CALIFORNIA—*con.*	ft. p. m.	ft.	ft.
Tuolumne.				
Columbia	Tuolumne	11 to 32	7½	4⅓
	COLORADO.			
Lake.				
California	Stevens and Leiter ...	5	10	3
Routt.				
Hahn's Peak	Elk River	10	4	3
Summit.				
	Boston G. and S. M. Co.	7	2½	2
	Gold Run	6⅔	5	1
	GEORGIA.			
Dawson.				
Fourth, Fifth, and Thirteenth	Anicola and Etowah ...	8	5	4
Lumpkin.				
Twelfth	Dahlonega	4	2	1
,, ...	Singleton	8	2	1½
White.				
Third	Nacoochee	5	5	3
Fourth	Town Creek	5	3	2
	IDAHO.			
Boise.				
Boston	Spencer	13½	2½	3
Moore's Creek	Alderson...	13½	4	2½
,, ,, ...	Christy	10	4	2½
,, ,, ...	Crepisculla (feeders) ...	10	2	1⅔
,, ,, ...	,, (main) ...	10	2½	2
,, ,, ...	Cuddy	13½	2	2
,, ,, ...	Dunn	13⅓	3	2½
,, ,, ...	East Moore's Creek ...	10	4	2½
,, ,, ...	Hanson (upper)... ...	10	3	2½
,, ,, ...	,, (lower)... ...	10	4	2½
,, ,, ...	Lambing...	13½	4	3
,, ,, ...	Mountain	10	3	2½
,, ,, ...	Plowman	13½	3½	2½
,, ,, ...	Thorn Creek	10 to 13½	4	2½
,, ,, ...	Upper Alderson ...	13½	3	2½
Pioneer	Stevenson and Noble ...	13½ to 16⅔	4	2
,, ...	Wilson	10 to 13½	4	2
	MONTANA.			
Beaver Head.				
Bannock...	Bannock D. and M. Co.		3	1½
,, ...	White Bar	6⅔	3½	1⅓

County and District.	Line.	Grade.	Width.	Depth.
	MONTANA—*con.*	ft. p. m.	ft.	ft.
Deer Lodge.				
Henderson Gulch ...	Smart's Creek	6⅔	6	2
Squaw Gulch ...	Rock Creek	10	7	3½
Meagher.				
German	Diamond Flume and Hydraulic Co.	13¾	5	3½

PLACER MINES.

Baker.	OREGON.			
Blue Cañon	Littlefield & Duckworth	13½	2	2
,, ...	Marysville	13½	4	2½
Chicken Creek	J. N. Osborne and Co....	6⅔	1⅔	1⅔
Clarksville	Placer G. M. Co. (Burnt River line)	6⅔	6	2½
,, ...	Placer G. M. Co. (Clark's Creek)...	20	4	2
Pocahontas	Baisly	20	3	2
Sumter	Young and Rimbol ...	9½	4	3
Coos.				
	Pioneer	7½	4	1½
Douglas.				
Big Bend Cow Creek ...	J. G. Fuller	17¾	1½	1¾
,, ,, ...	D. A. Leavans	13⅓	2½	2
,, ,, ...	D. F. Salmon	10	3	2½
Cañonville	Catching Brothers ...	13⅓	3	1
Grant.				
Cañon City	Humboldt	20	3⅓	2
Granite	Klopp and Johnson ...	30	1⅚	1½
Marysville	Hillis	6⅔	4	2½
,, ...	Miner's	5⅓	2	1⅔
,, ...	Thompson's	45⅚	3	1⅓
Jackson.				
Applegate	Grand Applegate ...	20	4	4
,, ...	Squaw Lake	16	4	3
Farris Gulch	Farris Gulch	10	4	2½
Forty-nine	Ashland	53⅓	3	1½
Sam's Valley	Hays and Magruder ...	6⅔	2	1⅔
Josephine.				
Waldo	Desselles, No. 1 ...	6⅔	4	3
,, ...	,, No. 2 ...	6⅔	3	3
,, ...	Waldo	6⅔	5	4
Yank	Blue Gravel	26⅔	3	2½
	Josephine	13⅓	4	3

To determine what will be the flow of water in flumes, streams, and ditches, the following Table* has been prepared to determine the capacity of those of different sections when the fall per mile is known:

FLOW OF WATER PER SECOND IN RECTANGULAR FLUMES.

Fall per Mile.	*W., 1.5, D., .75.	W., 2, D., 1.	W., 2.5, D., 1.25.	W., 3, D., 1.5.	W. 4, D., 2.	W. 5, D., 2.5.	W., 6, D., 3.	W., 8, D., 4.	W., 10, D., 5.
ft.	c. ft.	c. ft.	c. ft.	c. ft.	c. ft.	c. ft.	c. ft.	c. ft.	c. ft.
1	.351	.77	1.43	2.33	5.12	9.57	19.49	35.61	66.54
2	.497	1.09	2.03	3.29	7.24	13.64	27.56	50.36	94 15
3	.608	1.34	2.48	4.03	8.87	16.58	33.76	61.76	115.31
4	.702	1.55	2.86	4 65	10.23	19.14	38.99	71.21	133.15
5	.785	1.73	3.20	5.20	11.44	21.41	43.59	79.62	148.86
6	.860	1.89	3.51	5.70	12.54	23.44	47.75	87.21	163 07
7	.929	2.04	3.71	6.15	13.54	25.32	51.58	94.20	176.14
8	.993	2.18	4.05	6.58	14.47	27.07	55.14	100.71	188.30
9	1.053	2.32	4.30	6.98	15.35	28.72	58.48	106.81	199.72
10	1.110	2.45	4.53	7.36	16.19	30.27	61.64	112.59	210.52
11	1.165	2.56	4.75	7.71	16.98	31.75	64.65	118 09	220.80
12	1.216	2.68	4.96	8.06	17.73	33.16	67.52	123.34	230.62

* W. and D. respectively represent width and depth in feet.

The cost of flumes per yard differs very greatly, as they vary in size from a simple miner's launder set up on poles to those which are 6 ft. or 8 ft. in width and depth. The lowest cost per yard reported is 29 cents per yard in Clay County, Alabama. The highest is $14 per yard at Columbia, Tuolumne County, California. One is reported at Plowman, Boise County, Idaho, that cost $12, one at Squaw Gulch, Deer Lodge County, Montana, that cost $10. A cost of $5, $6, and $7 is quite common, but the average of most of them will not be more than from $1 to $8. The difficulty of position more than anything else affects the cost. It is not so much the size of the flume as the place in which it must be built which influences the expense. It will be seen by comparison of these figures that although flumes cost more than a ditch excavated in moderately loose ground, they cost a great deal less than rock cuttings or excavations in earth that must be walled, and hence their adoption in so many cases. They, however, cost from 40 to 60 per cent. more than a ditch. The cost of repairs is in about the same proportion as long as the flume lasts, and rarely ever costs less

* Prod. Gold and Silver in the U.S., 1884, p. 571.

than 11 per cent. of the original expenditure. The ditch when completed, if well built and properly treated and cared for, grows tighter and needs less repair every year, while the flume constantly depreciates, and in a few years must be entirely replaced.

Wherever it is possible to avoid building a flume it should be done, both because it is liable to accident and to frequent repairs, the latter costing 90 per cent. more than the repairs to the same length of ditch, and also because under the best conditions, free from ordinary accidents, the flume will not last longer than ten to fifteen years. This is all the more true in California, since most of these flumes run dry during a certain number of months in the year; and it is a well-known fact that wood that is sometimes wet and sometimes dry will last a much shorter time than that which is always either wet or dry. The alternate expansions and contractions destroy the fibre of the wood, and start the nails and wedges. In order to avoid this it has been proposed to make the flumes of sheet iron. When timber is scarce and the price high they could be made as cheap or even cheaper than wood, and would be much more durable, but it is better to use pipes at once, notwithstanding the friction, than to use iron flumes, especially where the distance over which the water is to be conveyed is great. It would seem wiser in such a case to resurvey the ground and carry the water round in a ditch. There are points, however, where this cannot be done, but until timber becomes very scarce in California, the use of iron flumes will not be likely to become general, since they will have to be transported in pieces, and put up on the spot, and will necessitate the use of all the timber which is now used in supporting wooden flumes. They would besides have to be coated with tar and kept constantly painted. In most cases where it might be desirable to use them, they would, in the end, probably cost more than the ordinary flume.

VI. The Construction of Pipes.

The difficulties which attended in many cases the construction of ditches, flumes, and launders, and the ease and facility with which wrought-iron pipe had been used as a feed pipe, suggested its use as supply pipe for the mines. Iron pipe seems to have

been introduced in the course of the year 1853, when about 100 ft. of stove pipe was used to replace canvas hose, and to have been manufactured on a large scale for the purpose of carrying water in 1857, when a sheet-iron pipe 40 in. in diameter was laid across a valley near Smartsville. The comparative ease with which it could be carried across deep cañons as well as over hills, reaching the ground where the water was needed by traversing the least possible distance and without great expenditure, soon not only resulted in its general adoption for this purpose, but also showed that the water formerly used only in mining might be advantageously carried where it might be used as power as well. The result of its introduction was that instead of discharging a few hundred inches of water against a bank it is now possible to discharge 1000 in. per minute, equal to 1579 cubic feet, or even more, with a velocity of 125 ft. to 160 ft. per second, a force against which no bank can resist. But in order to make it most effectual, bank blasts of 50 to 100 tons of explosives are made so as to facilitate the cutting effect of the water.

The difficulties which at first prevented the general adoption of wrought-iron pipe for the transportation of water were first that no one had tried it, and second that every one to whom the suggestion was made was certain that sheet iron would not stand the pressure, or, if it did, that the pipes would leak, or that they would have to be so permanently put up that they could not be suitably adapted to the many accidents which occur in the change of position so often required in the exploitation of deep placer gravel. None of these objections, however, proved to be well founded.

Previous to the use of sheet-iron pipe on a large scale in California, it had generally been supposed that water to be carried without danger at great pressure would have to be carried in cast-iron pipes. The successful use of the feed pipes made of comparatively thin sheet iron led to experiments and then to its habitual use as a method of supply. The material of which this pipe is made is so light, is so easily transported, is subjected to so few accidents and to so little decay, is so easily repaired and loses so little water, that, like most

simple things in engineering which accomplish great results, it is a matter of surprise that it was not used long before. It must, however, be said that it did not suddenly come into use. The knowledge of what can be done with it and how to use it is the result of many years of experience, first in the hydraulic mines and then in the use of water for all other purposes.

These pipes are made of various sizes according to the quantity of water needed. They are usually 11 in., 15 in., 22 in., and 40 in. The sheet iron varies from No. 12 to No. 16 Birmingham gauge. The pipes are made of variable lengths, which are rivetted together when they are first placed in position on the ground. The length first adopted was the box, or 12 ft., but the sections are now usually made 15 ft. or 20 ft. long. When they were first introduced many devices were used for putting them together, some of which were flanges, rivets, and wires, all of which are still used in a small way to connect either the sections or lines of considerable length. When they pass over moderately level ground, one end is simply inserted into the end of the next piece, which is made larger for the purpose, without any other attempt at fastening. Such joints, when carefully made or when long enough, are sufficiently tight. In many cases they are subject to no leak, and will remain tight under a pressure of 150 lb. to 200 lb. to the square inch, as, from the form of the pipe, any small interstices that there may be at its end will soon be filled up by the floating matter which is kept in place by the pressure of the water. When higher pressures than 150 lb. to 200 lb. are required, the pipes must be rivetted, leaded, or bolted together by flanged joints. Pipes 12 in. in diameter are made of light iron rivetted together with rivets every inch where the edges join. The lengths are inserted together and are rivetted with rivets 3 in. apart. When the joints are not air-tight, they are made water-tight by throwing in a few bags of sawdust or a few shovelfuls of fine earth at the head of the pipes, which soon fills the interstices between them and stops the leaks. When very strong pipe is to be made, the longitudinal joints are double rivetted with rivets 1 in. apart in alternate rows, so that there is a ½ in. between each two rivets. Such pipes, properly covered with a mixture of coal tar

and asphalte, will last for many years. Painting the pipes with a brush is only a palliative, and can never be a perfect protection. They should be raised to the temperature of boiling tar and then immersed in a bath of it. This makes a perfect and a permanent protection. It is always necessary to test the tar before using it. If when cold it is found to be brittle it must be rejected.

Such pipe replaces almost altogether, in the west, the cast-iron pipes which are so much used in our eastern cities. These pipes are very light; when they are to be moved it is only necessary to cut off the rivets and take up the sections and place them in their new position. The possibility of moving the pipes with facility is of great importance, though not so necessary with the supply as with the feed pipes. When they are to be mended, the section is taken out and a new one put in. Extra sections are always kept on hand in order to have the least possible delay when repairs are to be made. These pipes stand a very heavy pressure. They deteriorate more frequently by collapse than by bursting. The general diameter of the main supply pipe is 30 in. or more. They are always used to cross valleys that would be too deep for flumes. It is sometimes not desirable to carry a flume across certain places, and also inexpedient to carry the pipe as a syphon. In such a case, as at the Columbia Hill diggings, the pipe is carried over on frames built like a flume with only a top board for a floor.

The Spring Valley Water Company, which supplies the city of San Francisco, has seventeen miles of 30-in. rivetted wrought-iron pipe, which carries the water across depressions of 150 ft., 200 ft., and 250 ft. in vertical height. A 30-in. pipe 14,000 ft. long was laid, in 1870, by the Spring Valley Mining Company, of Cherokee, Butte County, California, across a valley of the west branch of the North Fork of Feather River, which is nearly 1000 ft. deep. In such a case the water must be discharged into a head box at a point high enough above the discharge level to give pressure sufficient to overcome the friction. The friction in this pipe has been calculated to be equal to 20 ft. of hydrostatic pressure. The dimensions of this pipe are as follows:

	ft.	in.
Length of pipe	1400	0
Diameter of pipe	0	30
Vertical height above the lowest point ...	980	0
Vertical height where water is discharged above the lowest point	830	0
Pressure of water	150	0
Greatest height to which the water rose ...	50	0
Quantity of twenty-four hour inches supplied, 2210		

The usual dimensions of feed pipes are as follow:

Diameter of Pipe.	Pressure.	Number of Iron.	Thickness of Iron in Decimals of an Inch.
in.	ft.		
22	150	16	0.060
22	150 to 250	14	0.078
22	250 to 310	12	0.098
30	150	14	0.078
30	150 to 275	12	0.098
40	160	...	0.236

The iron used for these feed pipes generally varies from No. 16 to No. 11, according to the greater or less pressure; the very best iron must be used. The size of the pipe will depend upon the supply of water. With 1500 in. to 2000 in. of water a 22-in. pipe will be all that is required. Where the supply is as much as 3000 in., pipe 30 in. in diameter must be used. For a greater supply a larger pipe will have to be constructed.

The rivets which are used for No. 14 iron are made of wire $\frac{1}{4}$ in. in diameter, for No. 12 iron the same, for No. 11 iron $\frac{3}{16}$ in., and for No. 7 $\frac{3}{8}$ in. The pipe is made in pieces 23 ft. long, and the sheets are rivetted together by a steam rivetting machine.

The thickness of the iron is usually proportionate to the head of water and the diameter of the pipe. Pipes made of the different sizes iron given below will stand the following strain per sectional inch:

Number of Iron.	Made to Stand Strain per Sectional Inch. Pounds Avoirdupois.
12	7,000 to 9,000
12 to 9	9,000 ,, 12,000
9 to $\frac{5}{16}$-in.	12,000 ,, 14,000
$\frac{1}{4}$-in to $\frac{3}{8}$-in.	17,000 ,, 18,000

The following Table gives the thickness and weight of wrought iron used in making pipes :*

United States Gauge.			Birmingham Gauge.		
No. of Gauge.	Thickness.	Weight per Square Foot.	No. of Gauge.	Thickness.	Weight per Square Foot.
	in.	lb.		in.	lb.
10	0.101890	4.0884	10	0.134	5.3767
11	0.090742	3.6410	11	0.120	4.8150
12	0.080808	3.2424	12	0.109	4.3736
13	0.071961	2.8874	13	0.095	3.8119
14	0.064084	2.5714	14	0.083	3.3304
15	0.057068	2.2899	15	0.072	2.8890
16	0.050820	2.0392	16	0.065	2.6081
17	0.045257	1.8159	17	0.058	2.3272
18	0.040303	1.6172	18	0.049	1.9661
19	0.035890	1.4400	19	0.042	1.6852
20	0.031961	1.2824	20	0.035	1.4044
21	0.028462	1.1420	21	0.032	1.2840
22	0.025347	1.0170	22	0.028	1.1235
23	0.022571	0.9057	23	0.025	1.0031
24	0.020100	0.8065	24	0.022	0.8827
25	0.017900	0.7182	25	0.020	0.8025

The head of the water in pounds avoirdupois, multiplied by the diameter of the pipe in inches and divided by the co-efficients given in the Table at the foot of page 161, gives twice the thickness of the iron to be used. Allowance must be made for the security required, that is, if the breakage of the pipe will cause much damage it is advisable to lower the margin for greater safety. The diameters of the rivets usually used are :

Number of iron	18	16	14, 12, 11	10, 8, 7	$\frac{1}{4}$-in.	$\frac{5}{16}$-in.	$\frac{3}{8}$-in.
Corresponding diameter of rivets	$\frac{5}{32}$-in.	$\frac{5}{32}$-in.	$\frac{5}{16}$-in.	$\frac{3}{8}$-in.	$\frac{1}{2}$-in.	$\frac{5}{8}$-in.	$\frac{3}{4}$-in.

They are usually spaced to make the pipe tight, that is, closer than is necessary for the strength of the seam ; but this, in turn, is governed by the pressure on the pipes.

The following Table shows the details of construction for a 22-in. pipe :†

* Report of U.S. Census, 1880, vol. xiii., p. 193.

† *Ibid.*, p. 233.

Thickness of Iron.	Diameter of Rivets.	Length of Rivets.	Pitch of Circle Seams.	Number of Rivets in each Circle Seam.	Pitch of Rivets in the Longitudinal Seam or Double Row.	Width between Centres of Rivets in the Double Row.
	in.	in.	in.	in.	in.	in.
No. 12	$0\frac{3}{16}$	$0\frac{3}{8}$	1	69	$1\frac{3}{8}$	$0\frac{5}{8}$
No. 11	$0\frac{3}{16}$	$0\frac{3}{8}$	1	69	$1\frac{3}{8}$	$0\frac{5}{8}$
No. 9	$0\frac{1}{4}$	$0\frac{13}{16}$	$1\frac{3}{16}$	59	$1\frac{7}{16}$	$0\frac{5}{8}$
$\frac{3}{16}$-in.	$0\frac{1}{2}$	$1\frac{1}{4}$	$1\frac{7}{16}$	39	$1\frac{11}{16}$	$1\frac{1}{16}$
$\frac{1}{4}$-in.	$0\frac{1}{2}$	$1\frac{1}{2}$	$1\frac{7}{16}$	39	$1\frac{11}{16}$	$1\frac{1}{16}$

STENGTH OF IRON PIPES.*

Number of Sheet Iron.	Thickness in Inches.	Pressure in Feet.	Pressure in Pounds.	Tensile Strain per Square Inch.
				lb.
14	.083	170	74	13,374
12	.109	288	125	17,202
11	.012	293	127	15,875
10	.134	355	154	17,240
$\frac{3}{16}$	.187	435	188	15,080
$\frac{1}{4}$	.250	594	257	15,420
$\frac{5}{16}$	.312	842	365	17,549
$\frac{3}{8}$	.375	887	384	15,360

The supply pipe discharges into a strong head box, as shown in Fig. 16, which is usually about 8 ft. deep. An iron grating should be placed at this point to keep out any floating materials. There should be about 5 ft. or 6 ft. of water over the outlet pipe.

The pipes of the Spring Valley Mining Company at Cherokee, Butte County, have a diameter of 30 in. and 34 in. The iron used was ordinary English plate of fair quality. The greatest pressure it sustains is 887 ft., and the thickness of the iron at that point is $\frac{3}{8}$ in. The pressure and maximum strains on the several sizes of iron used are given in the Table on the next page. The longitudinal joints are double-rivetted; the circular joints are double-rivetted and caulked. The longitudinal seams are double-rivetted; the circular seams are single-rivetted. Air valves are placed at several points.

The cost of these pipes per yard varies from $25 in difficult positions, as at Cherokee, Butte County, California, to 34 cents

* Trans. Am. Inst. Min. Eng., vol. vi., p. 71.

at McAdam's Creek, Siskiou County, California. At Mokelumne Hill, Calaveras County, it was $12; at Georgetown, El Dorado County, it was as high as $9, as also at Lumpkin County, Georgia. It varies generally between $1.50 and $3.50.

DETAILS OF THE SPRING VALLEY PIPES.*

SIZE OF IRON.		GREATEST PRESSURE.		Maximum Tensile Strain per Square Inch, in Pounds.
No. of Gauge (Birmingham Gauge).	Thickness in Decimals of an Inch.	Feet.	Pounds.	
14	0.083	170	74	13,374
12	0.109	288	125	17,202
11	0.012	293	127	15,875
10	0.134	355	154	17,240
3/16	0.187	435	188	15,080
1/4	0.250	594	257	15,420
5/16	0.312	842	365	17,549
3/8	0.375	887	384	15,360

When iron pipes used to carry water cross cañons, gorges, or depressions of the ground too deep to permit the construction of wooden flumes, inverted syphons are used. It is then necessary that the ditch be high enough at the point where the water is discharged into the pipe to allow a sufficient "head," not only to overcome the friction, but also to secure enough hydrostatic pressure to force the water rapidly through the pipe, which, of necessity, must be of far smaller dimensions than the discharging ditch.

As the efficiency of the work done depends on both the weight and the velocity of the water discharged from the nozzle which strikes the bank, it is desirable that the supply to the feed box come from a feed or reservoir at as high an elevation as possible and as near as convenient to the place where the water is to be used. This will give the greatest amount of pressure on the shortest line of pipe. Where six or more nozzles are working at the same time, as shown in Fig. 25, both the position of the supply and the arrangement of the pipes used needs the most careful study.

The feed box, Fig. 16, should have an overflow which dis-

* Report of the U.S. Census, 1880, vol. xiii., p. 233.

charges on to rock or on to a carefully laid pavement, so as to do no damage to the surrounding ground. From the head box the water is discharged into the feed pipes, which should go to the workings in as direct a line as possible, and, if possible, without bends and upon an even grade.

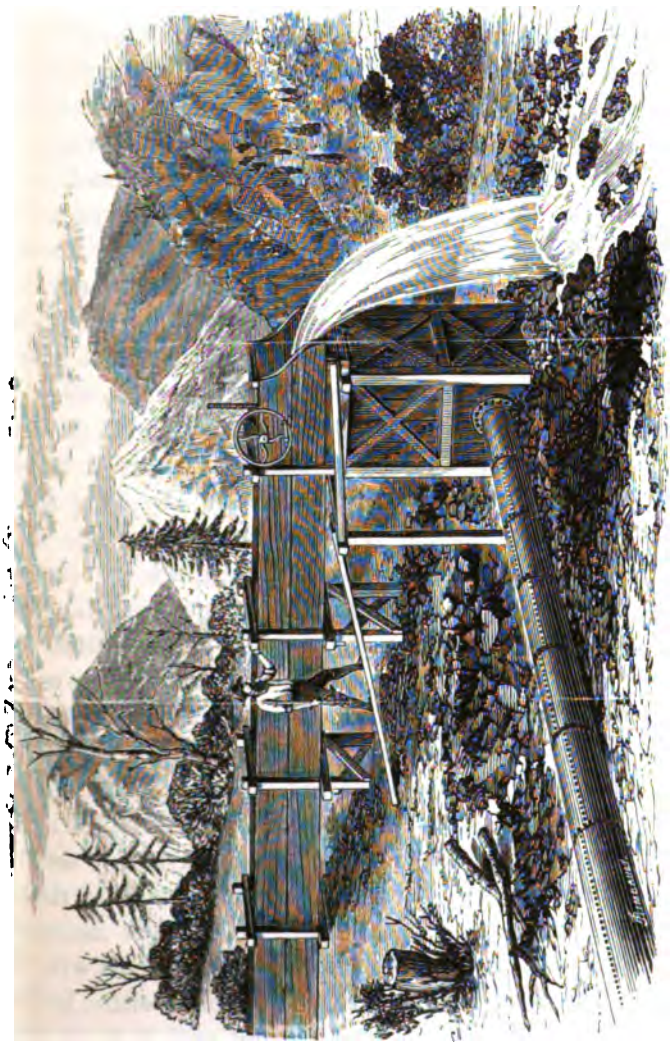


FIG. 16. THE HEAD BOX.

Before the pipe is put into position it should be provided about every hundred feet with air valves about 2 in. in diameter. These valves are of the greatest importance, since in case there

is any depression in the pipe, the air will be retained there, and if the water is discharged faster than it can enter the pipe, a collapse must take place unless air valves which can act quickly and allow the air to enter are in place to prevent a vacuum being formed. These valves are also useful when the pipe is being filled for the first time, or when for any reason it has been necessary to empty the pipe and it is refilled in order to use it. The valves should be made of leather weighted with brass and usually not with wood. The only object of wood is to cause the valves to close up themselves, but when wooden valves are used they have been known to swell and so change their form as not to act quickly enough, and collapses have taken place for that reason. These valves should open on the inside of the pipe and cover a hole from 1 in. to 4 in. in diameter. Sometimes a light spring which just counterpoises the weight is used to keep them closed, so that when there is the least diminution of pressure upon the inside they will open automatically. Sometimes the valves are made to rise and fall with the water in the pipes. They are thus closed automatically when it is full and open when it is partially empty. Every expedient should be adopted for preventing air entering the pipes. The water for this reason is made to enter either through a funnel-shaped pipe or by means of a gate which allows of regulating the supply at will. Sometimes a stand-pipe is put in at some little distance from the inlet, which catches the air as it floats along on the top of the water in the pipe and allows it to escape without doing damage. Collapses in these pipes occur more frequently than bursting. Every precaution should be taken to avoid them, for though the injured pipe can be quickly replaced, while it is being done the water must be cut off, and inconvenient delays may result. The deeper the water in the bulkhead over the mouth of the pipe, other things being equal, the less the amount of air which will enter it. This depth should be at least 4 ft. When the pipes run by steep declivities where there is either likely to be a constantly increasing pressure of snow, so that the pipes might be crushed by its weight, or when there is likely to be a strong movement of the snow which would be likely to displace the line of pipe, or where avalanches take place which might carry

the pipes away altogether, it must be protected by low sheds made of very strong and heavy timber.

These iron pipes not only carry the water on a general grade, but valleys and chasms over 1000 ft. deep are crossed by them, either spanned by wire cables or bridges. This can, however, only be done for a limited size of pipe. When the span is too great to be thus crossed inverted syphons are used which descend to the bottom of the valley and rise on the opposite side, to continue the grade from a lower level than the point from which the crossing was commenced. The line of the Virginia City and Gold Hill Water Company is one of the most remarkable. It includes an inverted wrought-iron syphon, 11½ in. in diameter, which sustains a pressure of 1720 ft. or 750 lb. per square inch at its greatest depression. At this point No. 0 iron is used, hot-rivetted with ⅝-in. rivets, with double rows on the longitudinal seam and single-rivetted on the round seam. It is said to have been tested to a pressure of 1400 lb. per square inch. A second syphon, made of lap-welded pipe, ¼-in. iron, 10 in. inner diameter, was subsequently added to the plant.

A pipe of the Spring Valley Water Company is 18 in. in diameter, 5800 ft. long, and has a tensile strain of from 5000 lb. to 6000 lb. per sectional inch. It is subjected to the pulsations of a single-acting plunger pump making at times thirty-six strokes per minute. At Cherokee, Butte County, the pipe carries the water across the West Branch, a tributary of the North Fork of Feather River. It is nearly 30 in. interior diameter, 14,000 ft. long, and at its lowest point is subjected to a pressure of 887 ft. It has a capacity of 52 cubic feet per second. The heaviest material used in its construction is ⅜-in boiler plate. It has been in use nearly ten years and shows but little wear. Where it crosses the Little Butte Creek it is about 1000 ft. long, and is depressed 148 ft.

The Vulture Pipe Line, Maricopa County, Arizona, is used for supplying the Vulture 80-stamp mill. It is sixteen miles long and the diameter for the first five and a half miles is 9 in.; and for the rest of the distance, 7¼ in. It is of No. 19 iron, in bands of 2½ ft., made into sections 24 ft. long. The longitudinal joints are slip-joints rivetted together, the circular joints are made by

slipping 4 in. of the end of one section into the next without rivetting, and are wrapped around with rags. There are no air-valves. For five miles the pipe falls 5 ft. to the mile, and water is then raised 320 ft. in a distance of 6200 ft. by steam pumps driven by a 50 horse-power compound engine; thence to point of delivery the grade is 11 ft. per mile. The greatest pressure is about 40 lb. per square inch.

The supply pipe ends in a distributor, from which several pipes of smaller size are used to carry the water to nozzles. The distributors supply the feed pipes. They often have four gates if the supply of water is sufficient to work four nozzles from one single box. The distributor is a box generally made of cast iron, and is provided with openings to which the pipes connected with the nozzles are attached. These openings are closed with ordinary water gates, which are manipulated by means of a screw upon the outside. The water is in this way turned on to or off from any one of the nozzles, so that the whole or any part of the water may be discharged from any single nozzle, or may be cut off from one or more. When the most convenient place for the distributor has been selected, it is firmly fastened there, to remain permanently.

The feed pipe which carries the water from the pressure box to the distributor and from there to the nozzles was first made of canvas hose. But this rots rapidly, and is liable to burst from any extra pressure of water, and then cannot be easily repaired, so it was not long before iron pipe very generally took its place. It was brought into general use about 1850, and for more than twenty years was used with great advantage. It was made of the best three-ply canvas, either single or double, and carefully sewed together. It was usually not more than 6 in. to 8 in. in diameter, and the size of the discharge nozzle used with it was about 2 in. As sailors usually sew, it will bear a pressure of 50 ft. to 100 ft. perpendicular height without any other support. When greater pressure is required it is surrounded by iron rings about 3 in. apart, each one being connected by cords. So arranged, it is called "crinoline" hose, and will bear a pressure of 200 ft. of water. When iron rings cannot be had the hose is covered over with a ne

made of rope ¼ in. to ½ in. in diameter, with meshes about 2 in. square, and is then almost as strong as the crinoline. This pipe was easily shortened, lengthened, repaired, and transported, and seemed at the time of its invention the perfection of water conduit; but in the general march of improvement, iron pipes took its place, and it has now been almost entirely superseded by them, though it is still used in some localities which are not easy of access, or upon very small claims, or where the pressure is not high, or where it would not be advisable to put up the plant necessary for the manufacture and use of iron pipes.

The pipes which lead from the distributor to supply the nozzles vary from 10 in. to 15 in. in diameter; they are made of No. 14 to No. 12 sheet iron, and are about 12 ft. in length. This length was that usually adopted for the sluice of early shallow placer mining, and is still called a "box." It is made up of pieces of sheet iron, about 18 in. long, rivetted together. The boxes are sometimes connected by flanges, or by hooks, and wire. At Smartsville bolts and nuts are used, so that when necessary the boxes can be easily separated. The joints in all cases must be made perfectly tight. In some places the end of one joint is made to fit into the end of the next. The boxes are thus put together like a stove pipe, and without any other support. This should not usually be done except for a very temporary construction. The line of pipe is generally placed on supports, as shown, Fig. 25, to avoid elevation and depression in the pipe, but if it is laid on the ground the air valves must be carefully looked after, and each box should be thoroughly coated with coal tar.

Angular beds in the pipe should be carefully avoided. When the direction must be changed, it should be done with curves of large radius. If a single sharp bend must be made, it will not perceptibly hinder the flow of the water if it does not exceed 4 ft. in 10 ft., and the velocity of the water is not greater than 6 ft. per second, or four miles an hour. When the bend is greater than this an additional head will have to be provided to get the same amount of water. The amount of this head will depend on the velocity of the water and the deflection of the pipe. Generally the velocity will not exceed 6 ft. per second, with which the additional head required will be 2 per cent. of the fall for a

deflection of 5 ft. in 10 ft., and 10 per cent. for a deflection 10 ft. in 10 ft.

The miners always repair their own pipes where they have been injured by collapse or any other cause; the box is taken out and carried to the shop, where the injured piece is removed. If it is too much damaged to be used again, a new piece is put in, if not, the rivets are removed, the sheet straightened between rollers, made square with shears, and replaced. If only a hole has been made, a piece of sheet iron is rivetted over it. On a large claim there will always be a number of men engaged in this kind of repair.

The use of single-feed pipes in the larger works is being rapidly abandoned. To prevent rusting the pipes are painted with a mixture of asphalte and coal tar. When the pipes are placed in position where they are likely to remain permanently for some time they are placed on framework, as shown in Fig. 25. It has been estimated that the aggregate length of such feed pipes in hydraulic mines in the United States is over 200 miles. The pressure of the water varies up to 200 ft. Every ditch company tries to bring its water to the highest possible elevation so as to command the greatest area. As previously remarked, of two companies with otherwise equal facilities the one which commands the highest elevation commands the highest price for the water. A straight line connecting the source of supply with the outlet of the water is called the "hydraulic gradient." If there is any obstruction on the line then the hydraulic gradient for that line is the line connecting the source of supply with the highest point of the obstruction. The vertical distances between any part of the pipe and the hydraulic gradient multiplied by .434 gives the pressure in pounds per square inch at that point. The following Table* gives some of the dimensions of the principal pipes in the United States.†

* Placer Mines and Mining Ditches, U.S. Census, vol. xiii., p. 190.

† The values of the Birmingham and United States gauges in fractions of an inch have been given on p. 162.

DETAILS OF FEED PIPES.
CALIFORNIA.

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
		ft.		ft.	in.	
Butte.						
Cherokee Flat	Spring Valley	311	3	...	15 and 22	$\frac{1}{32}$ in., $\frac{1}{14}$ in., $\frac{1}{16}$ in.
Calaveras.						
Mokelumne Hill	Eureka	400	2	...	11 and 13	Nos. 14 and 16, U.S. gauge.
Del Norte.						
	China Creek	225	2	2680	11 and 15	No. 16.
	Del Norte	100 to 175	3	4000	11, 15, and 19	No. 16.
	Happy Camp	125 to 200	2	3100	11 to 24	No. 16.
	Muc-a-muc	90	1	1100	11 to 15	No. 16.
	Wingate	260	1	1200	15	No. 16.
Placer.						
Dutch Flat	Polar Star	450	2	...	Tapering from 4 ft. to 11 in.	Nos. 14 and 16.
Gold Run	Gold Run	400	3	...	,, ,, 5 ft. to 16 in.	No. 14 Birm. gauge.
Iowa Hill	Orion	440	1	...	15 to 22	Nos. 18 to 12.
Plumas.						
	Plumas M. and W. Co.	350	5	...	15 and 22	Nos. 16, 14, 12.
Siskiyou.						
Indian Creek	Bay City...	200 to 300	1	1000	8	No. 20.
,, ...	Coyote Gulch*	184	2	900	6	No. 18.
McAdam's Creek	Hiyou	150 to 200	1	800	6	No. 20.
Oro Fino	John Young	200	1	2000	6	Nos. 20 and 24.

* Galvanised sheet iron.

CALIFORNIA—*continued.*

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
		ft.		ft.	in.	
SISKIYOU—*continued.*						
Sciad Valley	Fort Goff Creek ...	275	1	866	11 to 15	No. 16.
,, ...	Thompson Creek ...	65	1	800	15	No. 16.
STANISLAUS.						
La Grange	La Grange	20 to 125	...	...	Main, 22 ; others, 15 to 36	
TRINITY.						
Buckeye	Buckeye	300	...	2073	11 to 15	
Red Hill	McKenna	290	...	2100	7 and 15	No. 16.
Weaver	Centre Placer	250	...	1400	13	No. 16.
	Trinity	270	...	4500	15	Nos. 14 and 16.
	COLORADO.					
CLEAR CREEK.						
Jackson	Edward	80	1	...	8 to 24	No. 18.
LAKE.						
California	Ore M. D. and F. Co. ...	140	...	...	8 to 30	No. 14.
PARK.						
Alma Township	Alma	80	2	...	16 to 22	No. 20.
ROUTT.						
Hahn's Peak	Hahn's Peak	225	1	...	15	$\frac{1}{16}$ in.

COLORADO—*continued.*

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
		ft.		ft.	in.	
SUMMIT.						
Spaulding	Rob Roy...	85	...	...	18	
DAKOTA.						
PENNINGTON.						
Rockerville	Black Hills Placer Co....	200	...	...	9 to 28	$\frac{1}{8}$ in. and $\frac{3}{8}$ in.
GEORGIA.						
HALL.						
810 G. M.	Glade*	156	...	900	6	$\frac{1}{8}$ in.
LUMPKIN.						
Twelfth	Battle Branch	150	1	...	9	No. 16, wire gauge.
,, ...	Findley*...	...	1	...	4	No. 16, ,, ,,
,, ...	Hand and Barlow ...	105	1	520	6 to 16	
UNION.						
Tenth and First	Coosa†	110	1	...	6	
IDAHO.						
BOISE.						
Moore's Creek	Noble, Lower, & Mann	114	...	920	7 to 11	Nos. 18 and 20.
Pioneer	Stevenson and Noble ...	150 to 200	...	...	11 to 20	Nos. 16 and 18.
MONTANA.						
DEER LODGE.						
Gold Hill	Squaw Gulch	130	4	...	11 and 15	$\frac{1}{8}$ in.
Henderson Gulch... ...	Butler and Anderson ...	125	1	...	6	$\frac{3}{32}$ in.

* Galvanised sheet iron. † Rubber.

MONTANA—*continued.*

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
DEER LODGE—*con.*		ft.		ft.	in.	
Independence	Oro Fino	60	1	1500	6 and 7	Nos. 18 and 20.
Nelson	Pioneer*...	110 to 150	1	...	8	$\frac{1}{16}$ in.
MEAGHER.						
German	Diamond Flume and Hydraulic...	200	1	...	11	$\frac{1}{16}$ in.
Thompson Gulch	California	60 to 75	1	...	6	$\frac{1}{32}$ in.
NORTH CAROLINA.						
MONTGOMERY.						
El Dorado	Beaver Dam	25	1	...	5	$\frac{1}{4}$ in.
POLK.						
	Cochrane	50	1	...	5	No. 20, Birm. gauge.
OREGON.						
BAKER.						
Humboldt Basin	Colt and Co.	60, 80, and 175	3	...	7	No. 20.
Pocahontas...	Baisly	300	1	...	11	$\frac{1}{18}$ in.
,, ...	Baisly and Daly ...	300	1	...	11	$\frac{1}{18}$ in.
,, ...	Salmon Creek	300	2	...	11 and 13	Nos. 18 and 16.
Rye Valley	Powers and Co.... ...	90, 100, and 140	5	...	8	$\frac{1}{18}$ in.
Shasta	Campbell	146	...	800	8	No. 20.
,, ...	Steiger's French Gulch...	100	1	...	7	$\frac{1}{18}$ in.

* Canvas.

OREGON—*continued.*

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
DOUGLAS.		ft.		ft.	in.	
Big Bend Cow Creek ...	J. Fuller...	200	...	700	11	$\frac{3}{64}$ in.
,, ,, ...	Merriam and Anderson	160	1	1400	7 to 10	$\frac{1}{16}$ in.
,, ,, ...	Salmon	175	...	1000	15	No. 16.
Cañonville	McWilliams and Ash ...	160	1	800	11	$\frac{3}{64}$ in.
Green Mountain	Charles Dwelley ...	200	1	300	5	$\frac{1}{16}$ in.
,, ,, ...	M. T. Dyer and Son ...	125	2	750	5 and 7	$\frac{1}{16}$ in.
GRANT.						
Marysville	Cannon and Co. ...	200	1	...	11	$\frac{1}{18}$ in.
,, ...	Thompson	140 to 230	2	...	11	$\frac{1}{18}$ in.
Rock Creek	Erickson Owens ...	150	1	...	7	
,, ...	Lasswell and Co. ...	200	1	...	7	
JACKSON.						
Applegate	Squaw Lake	327	1	1900	15 to 22	$\frac{1}{16}$ in.
Coyote Creek	Wm. Ruble	200	...	700	11	$\frac{3}{64}$ in.
Farris Gulch	Farris Gulch	300	2	1200	7	$\frac{3}{64}$ in.
Forty-nine...	Davenport	100	...	1120	11 to 22	$\frac{3}{64}$ in.
,, ...	Forty-nine	150	...	1100	6 to 10	$\frac{3}{64}$ in.
Sterling	Sterling	330	2	...	16 to 22	$\frac{1}{16}$ in.
Uniontown...	Chapel and Co.	300	1	...	15	No. 16.
,, ...	Gin Lin	250	2	...	15	$\frac{1}{16}$ in.
Wolf Creek	Gross Brothers	100	1	600	11	$\frac{5}{64}$ in.
JOSEPHINE.						
Grass Creek	Goff, Triplett, & Chapin	150	1	630	7 to 11	$\frac{3}{64}$ in.
,, ...	Steam Beer	150	1	1400	11	$\frac{3}{64}$ in.

OREGON—*continued.*

County and District.	Mine.	Head Pressure.	Number in Use.	Length.	Diameter.	Thickness.
JOSEPHINE—*con.*		ft.		ft.	in.	
Waldo	Deselles and Co. ...	300	2	2800	6 to 11	Nos. 16 and 18.
,, ...	Waldo	150	1	900	13 to 22	$\frac{3}{64}$ in.
Jank	Blue Gravel	160	3	1050	16	No. 16.
,, ...	Josephine	180	2	1500	11	No. 16.
SOUTH CAROLINA.						
CHESTERFIELD.	Brewer*	65	2	1700	5	No. 22.
UTAH.						
SALT LAKE.						
West Mountain	Argonaut	100	...	...	10	$\frac{3}{32}$ in.

* Galvanised sheet iron.

DETAILS OF THE CONSTRUCTION OF PIPES.*

County and District.	Mine.	Pressure.	Depth.	Diameter.	Joints.	Air Valves.	Thickness.
		ARIZONA.					
YAVAPAI.		ft.	ft.				
	Jackson ...	...	...	For 200 yds., 1 ft.; for 175 yds., 7 in.; balance, 7 in.	Longitudinal joints are lapped and rivetted; circular joints lapped 2 in. and held together by wiring four ears on each joint. These ears are rivetted to the pipe.		
		CALIFORNIA.					
BUTTE.							
Morris Ravine...	Morris Ravine	...	...	22 in.	Single rivetted; tarred before joining.	Placed where the pipe changes its angle.	Nos. 12, 14, and 16 at depths of 250 ft. to 310 ft., 150 ft. to 250 ft. and 150 ft. respectively.
CALAVERAS.							
Mokelumne Hill	Cooke & Clarke	...	...	12 in.	All rivetted.	One; near head.	Under greatest head No. 14 Birmingham gauge.
,, ,,	Mokelumne and Campo Seco...	...	70	12. in to 14 in.	Sections (18 ft. or 20 ft.) stove-piped and rivetted.	... ...	Nos. 16 and 14 Birmingham gauge.
EL DORADO.							
Georgetown ...	California ...	...	40-200	7½ in. to 22 in.	Longitudinal slip joints stove-piped and rivetted; circular joints rivetted.	None.	Nos. 12, 16, 18, and 20.
,, ...	El Dorado ...	...	...	16 in. to 30 in.	20-ft. sections; slip joints rivetted.	Every 100 ft.	Nos. 18 to 12 Birmingham gauge.

* Report U.S. Census, 1880, vol. xiii., p. 233.

CALIFORNIA—*continued.*

County and District.	Mine.	Pressure.	Depth.	Diameter.	Joints.	Air Valves.	Thickness.
EL DORADO—*con.*		ft.	ft.				
Georgetown ...	Park Canal ...	...	128	15 in. to 22 in.	Longitudinal joints stove-piped and wired or double rivetted; circular joints rivetted.	At highest points.	Nos. 18 to 14 Birmingham gauge.
PLACER.							
Gold Run ...	Gold Run ...	...	...	16 in.	Joints stove-piped and rivetted.	150 ft. apart throughout length of pipes.	No. 14 Birmingham gauge throughout.
PLUMAS.							
Seneca	Dutch Hill ...	...	...	22 in.	Longitudinal joints single or double rivetted according to pressure; circular joints are driven together and fastened by means of hooks.	Placed at the highest point along the line.	Varies between single rivetted No. 14 to double rivetted No. 12, the latter used at point of greatest pressure, 518 ft. five miles from the head of the pipe.
TUOLUMNE.							
Columbia ...	Tuolumne ...	...	...	6 in. to 16 in., averaging 11 in.	Rivetted.	At head.	Average No. 16 Birm. gauge; 0.065 in. shell weighing 2.627 lb. per square foot.
COLORADO.							
LAKE.							
California ...	Stevens & Leiter	...	...	13 in., 15 in., 18 in., 21 in., and 30 in.	... ...	None.	All No. 14.

DAKOTA.

County and District.	Mine.	Pressure.	Depth.	Diameter.	Joints.	Air Valves.	Thickness.
		ft.	ft.				
LAWRENCE. Whitewood ...	Black Hill Canal and Water Co.	...	250	8 in.	Lap joints held by four lugs drawn together by wire	... ...	No. 14.
GEORGIA.							
LUMPKIN. Twelfth	Dahlonega ...	...	...	12 in.	Spiral pipes, single rivetted. Circular sleeve joints, ends fluted and joined by wired lugs.	None.	No. 16.
,, ...	Hand*... ...	...	...	36 in.	Longitudinal lapped joints, double rivetted in one section, 42 ft. in length, single rivetted in the other sections. Circular flange joints made of cast iron turned in lathe, made tight with lead and bolted with $\frac{3}{4}$-in. bolts at intervals of 6 in. in the circumference. Joints rivetted to the pipes.	None, but the pipes are provided with man holes.	At greatest pressure $\frac{3}{8}$ in.; at other points $\frac{5}{16}$ in.

* Grade between head and discharge, 8 ft. to the mile.

GEORGIA—*continued.*

County and District.	Mine.	Pressure.	Depth.	Diameter.	Joints.	Air Valves.	Thickness.
WHITE.		ft.	ft.				
Fourth	Town Creek*...	...	...	Of upper third, 20 in.; middle third, 18 in.; lower third, 16 in.	Circular sleeve joints, one end corrugated, united by lugs and wire. Joints covered over with rubber cloth and three thicknesses of tarred canvas, with three iron clamps over each joint.	Used.	$\frac{3}{32}$ in.
IDAHO.							
BOISE.							
Moore's Creek ...	Plowman ...	84	...	11 in.	Longitudinal slip joints; circular joints made with an elbow in one piece.	None.	No. 18.
MONTANA.							
MEAGHER.							
German... ...	Diamond ...	440	440	22 in.	Rivetted.	None.	$\frac{1}{4}$ in. throughout.
OREGON.							
GRANT.							
Granite	Klopp & Johnson	...	Grtst., 84	18 in.	Longitudinal joints rivetted; circular slip joints.	None.	$\frac{1}{16}$ in. throughout.
JACKSON.							
Farris Gulch ...	Farris Gulch ...	60	...	22 in.	Longitudinal slip joints; circular joints made with curved elbow, rivetted.	Three near head.	$\frac{1}{16}$ in. ,,

* Spiral pipe, single rivetted.

The following Table* gives the flow of water per second in clean iron pipes. When the pipes are foul the quantities given in the Table must be reduced from .1 to .3 according as the pipes are more or less foul.

Fall per Mile.	6 in. Diam.	9 in. Diam.	11 in. Diam.	15 in. Diam.	22 in. Diam.	30 in. Diam.	36 in. Diam.	44 in. Diam.
ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.
1.58	...	...	...	...	...	7.78	12.70	22.22
2.11	...	...	...	...	...	8.99	14.56	25.55
2.64	...	...	...	...	...	10.24	16.35	28.87
3.17	...	...	...	...	4.61	10.97	18.02	31.46
3.70	...	...	...	...	5.25	11.90	19.76	34.47
4.22	...	...	...	2.05	5.62	12.84	20.85	37.05
4.75	...	...	...	2.19	6.01	13.48	22.30	39.01
5.28	...	...	...	2.30	6.32	14.21	23.47	41.06
5.81	...	...	...	2.43	6.62	15.05	24.91	42.09
6.34	...	...	1.12	2.54	6.94	15.81	26.12	44.97
6.86	...	...	1.17	2.65	7.24	16.47	27.20	46.77
7.39	...	...	1.22	2.75	7.51	17.81	28.24	48.83
7.92	...	...	1.27	2.84	7.78	17.94	29.19	50.62
8.45	...	.779	1.32	2.94	8.05	18.58	30.29	52.46
8.98	...	.803	1.36	3.08	8.36	19.21	31.42	54.04
9.50	...	.827	1.40	3.11	8.55	19.66	32.48	55.48
10.03	...	.851	1.45	3.21	8.85	20.32	33.40	57.01
10.56	.298	.875	1.49	3.29	9.07	20.79	34.49	58.85
11.62	.314	.917	1.58	3.47	9.55	21.80	36.15	61.71
12.67	.330	.960	1.65	3.63	10.01	22.83	37.74	64.35
13.73	.346	1.00	1.72	3.79	10.48	23.93	39.40	66.87
14.78	.359	1.04	1.79	3.95	10.91	24.86	40.86	69.57
15.84	.377	1.09	1.85	4.11	11.29	25.87	42.88	72.32
18.48	.395	1.17	2.00	4.46	12.25	27.96	45.95	77.95
21.12	.444	1.27	2.14	4.78	13.12	29.84	48.83	83.60
26.40	.496	1.42	2.40	5.37	14.78	33.55	54.89	93.37
31.68	.548	1.56	2.64	5.91	16.20	36.79	59.95	103.28
36.96	.589	1.69	2.86	6.46	17.53	39.66	65.17	111.74
42.24	.631	1.81	3.06	6.90	18.78	42.39	69.80	119.93
47.57	.672	1.92	3.23	7.31	19.93	45.23	74.33	128.26
52.80	.721	2.03	3.42	7.70	21.06	47.71	78.46	
63.36	.784	2.24	3.76	8.31	23.07	52.91	82.84	
73.92	.858	2.43	4.02	9.15	24.68	57.65		
84.48	.922	2.61	4.39	9.81	26.97			
95.04	.975	2.77	4.68	10.47	29.70			
105.60	1.02	2.93	5.25	11.09	31.15			
158.40	1.26	3.63	6.09	13.66				
211.20	1.48	4.22	7.02	15.84				
264.00	1.67	4.76	8.24					
316.80	1.93	5.24						
369.00	1.98	5.69						

The proportionate length of the ditches, flumes, and pipes used in connection with mines actually reported to the United States Census† is given in the Table on next page.

* Prod. of Precious Metals in the U.S., 1884, p. 571.

† Report of U.S. Census, 1880, p. 210.

	Miles.	Percentage of Length.
Total length of flumes reported on ...	135.59	4.44
,, ,, pipes ,, ,, ...	50.53	1.66
,, ,, ditches proper reported on	2863.38	93.90
Total length of ditch lines ...	3049.50	100.00

The total length of flumes is estimated at 450 miles, and of pipes at 150 miles, which are included in the general estimate of the total length, which is 10,873 miles. It will be seen* that by far the greatest amount of this length is comprised in the State of California, and that it comprises six-tenths of all the ditch lines of the country. The length, however, does not fairly represent their importance, as their individual capacity is much greater in California than it is elsewhere. No general† estimate can be made as to the entire amount of work which has been done in the way of ditch construction since the beginning of hydraulic mining. Thousands of miles of now idle and useless ditches were built in camps long since deserted; and in localities where placer mining is still profitably prosecuted there are also many idle ditches which have been supplanted by newer and larger ones at higher grades. As an instance in point, the case of Boisé Basin, Idaho, may be cited. In this one district alone there are estimated to be no less than 800 miles of old and abandoned ditches, which cost about $850,000, and aggregated more than double the length of the lines now in operation. Many such cases exist in California. Even a very rough approximation to the extent of these old, disused lines throughout the country is hardly practicable; but it can only be broadly stated as being very large.

VII. The Use of Nozzles.

In order to discharge the water against the bank with sufficient force a number of nozzles called "giants" or "monitors" are employed, some of which are of very ingenious construction. The first of these nozzles was an ordinary tin tube with an inch opening which was held in the hand like an ordinary hose nozzle. This tube was gradually increased in size, when it was found that its want of flexibility was a very great disadvantage. It was superseded by a pipe attached to a very short length of canvas hose, which was called a "goose-neck,"

* See p. 135. † Report of U.S. Census, 1880, p. 210.

Fig. 17, which consisted of two elbows of iron pipe A and A', the one movable on the other, with a horizontal joint at B between them. Beyond the upper elbow was a short piece of flexible hose E, and then the nozzle proper C. The joint B gave a movement of rotation in a horizontal plane; the hose allowed of a certain amount of movement in any plane. This was a great improvement over the straight nozzle, since it allowed easy movement in every direction; but the abrupt double curve of the elbow diminished the force of the water, and the upward pressure made the joint B work hard, and required precautions to prevent its "bucking," which is the description given by the miners to the sudden and almost irresistible movement in an opposite direction which often occurs from too great horizontal or vertical movement in the hose pipe, and frequently results in serious accidents to those who are not familiar with this peculiarity of the nozzle.

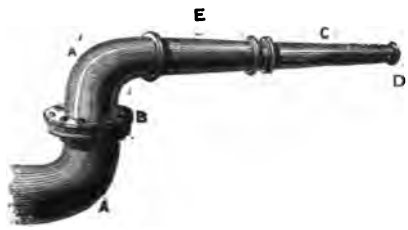


FIG. 17. THE GOOSE NECK.

A decided improvement upon this was "Craig's globe monitor,"

FIG. 18. CRAIG'S GLOBE MONITOR.

Fig. 18, which was formerly used at Smartsville, North Bloomfield, and elsewhere. This consisted at first simply of a hollow globe H, with an opening at the side into which the main supply pipe feeds, and an opening in the top for the nozzle. This has

an elbow with a hemispherical attachment J, which is turned so as to exactly fit the interior of a flange K, which is bolted to the globe, and makes, with a leather packing, a water-tight ball-and-socket joint. This permits the movement of the nozzle in any direction at pleasure, allowing of a vertical motion of nearly 40 deg., and of a horizontal motion of 360 deg. It was afterwards modified so that the strain on the joint was thrown by a lever F on to an interior bolt. Fixed in the elbow A is a wrought-iron tube C, 10 ft. long; to the end of this tube the nozzle proper D, which is of cast iron 6 in. to 8 in. inside diameter, is screwed. The whole nozzle was held firmly in position by being bolted to wooden beams L. It was usual, when the direction had been fixed, to tie up the nozzle, but when the pressure of the water was, say, 180 ft., it would remain where it is put without any support, although it was always safe to secure it.

The next machine used was called "Hoskins' dictator." It was invented in the year 1870, by R. Hoskins, of Dutch Flat, Placer County California. The main difference between it and the globe monitor being that the joint was external instead of internal, so that the water, instead of forcing the joints together as in the globe monitor, tended to force them apart. The joint was provided with an india-rubber packing. Friction wheels were introduced to diminish the friction, and as the pressure was all against elastic packings, the movements were easy, and the nozzle easily managed. It had all the motions of the globe monitor, but was more complicated in construction. There was a considerable loss of power from the abrupt curves of the joint; a defect which might easily be remedied. This nozzle was at one time extensively used.

The next one introduced was "Fisher's knuckle-joint and nozzle," Fig. 19, which is sometimes called "the hydraulic chief." It was invented by F. H. Fisher, of Nevada County, California. It consists of two elbows placed together in the form of an S, arranged in such a way as to revolve horizontally by means of a ring joint B provided with friction rollers. This ring is slipped down over the lower elbow, and is held to its place by a flange on the pipe. It was then bolted to a flange on the top elbow connecting the two, allowing the latter to move. The pressure of the water is such as to reduce the friction, so that it can be moved

easily. The joints are made water-tight by an india-rubber packing. The nozzle itself is fastened to the upper elbow by means of a knuckle-joint E. The vertical movement is effected

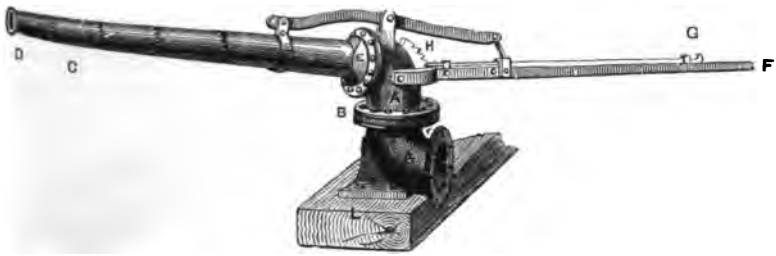


FIG. 19. FISHER'S KNUCKLE-JOINT AND NOZZLE.

by a series of levers G F, the main arm of which is 10 ft. to 12 ft. long, and is held in place by a ratchet. The movements in every direction are easy, and allow of an instantaneous change. The elbow and knuckle-joint are made of cast iron $\frac{5}{8}$ in. to $\frac{3}{4}$ in. thick. The pipe C is 8 ft. long of No. 16 sheet iron; the nozzle D, which is screwed on to the end of the pipe, is of cast iron. This machine has been improved as shown in Fig. 20, and is now

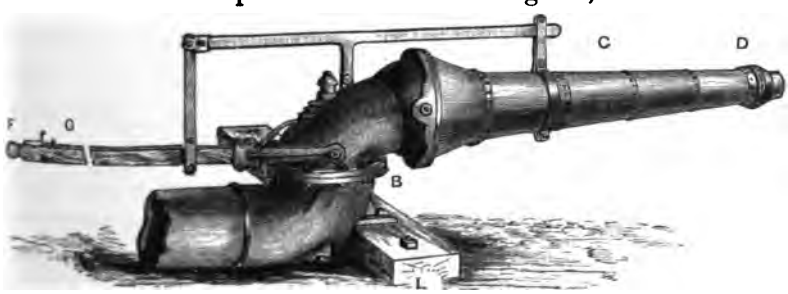


FIG. 20. FISHER'S DOUBLE-JOINTED GIANT.

known as "Fisher's double-jointed giant." The pipe when in use keeps its position by the force of the water, and with the lever can be easily directed to any point, and does effective work at a distance of 200 ft., and like all the modern giants secures the men from the danger of caves.

The nozzle which is now most used was invented by R. Hoskins, and is known as "the little giant." It was first invented and for a long time used as shown in Fig. 21, and a slightly different form of it in Fig. 22. It has a complete horizontal movement on the plane B B, which in this and all its

other forms is circular, and a vertical one on a knuckle-joint E, which is counterpoised at I to keep it in position. It is exceed-

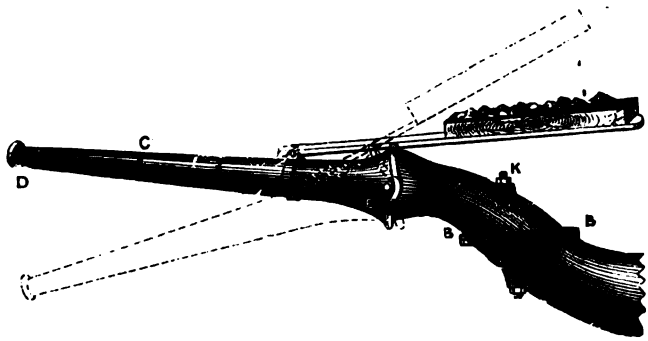


FIG. 21. THE LITTLE GIANT.

ingly simple and easily repaired. It is claimed for it, and its improved forms, that they discharge a larger amount of water

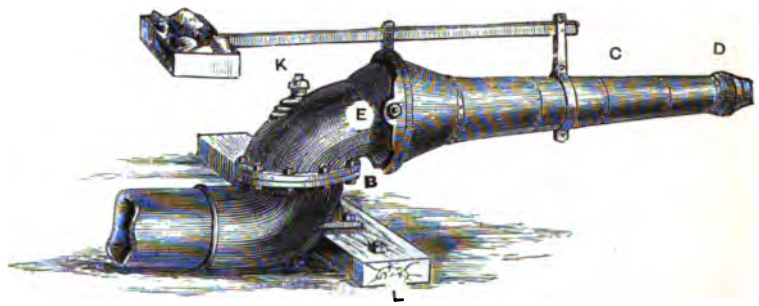


FIG. 22. THE HYDRAULIC GIANT.

with less resistance than any others. The joints are packed water-tight with leather or rubber, and are thus prevented from

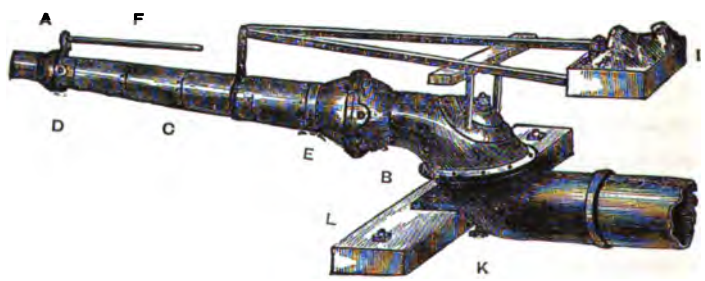


FIG. 23. THE HYDRAULIC GIANT.

wearing. Many years' experience with this nozzle has produced some simple changes in its form, which are shown in Figs. 22 and 23. In this last form it is known as "the hydraulic giant."

This giant is particularly adapted for very heavy pressures. Its construction is simple and it is not likely to get out of order. It has been used under pressures varying from 400 ft. to 535 ft. without accident.

Later R. Hoskins invented a nozzle known as "Hoskins' single-jointed giant," or "new giant," Fig. 24, which is much lighter than any of the others; the horizontal motion is in the joint which

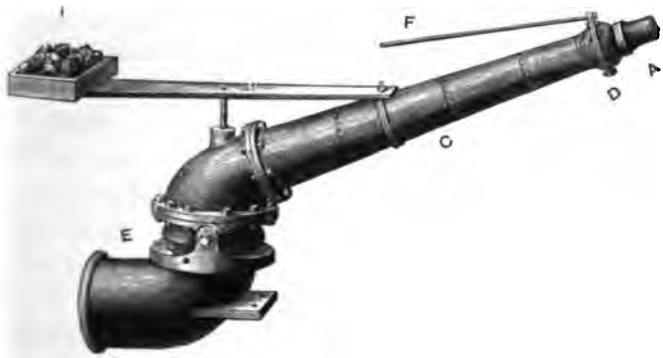


FIG. 24. HOSKINS' SINGLE-JOINTED GIANT.

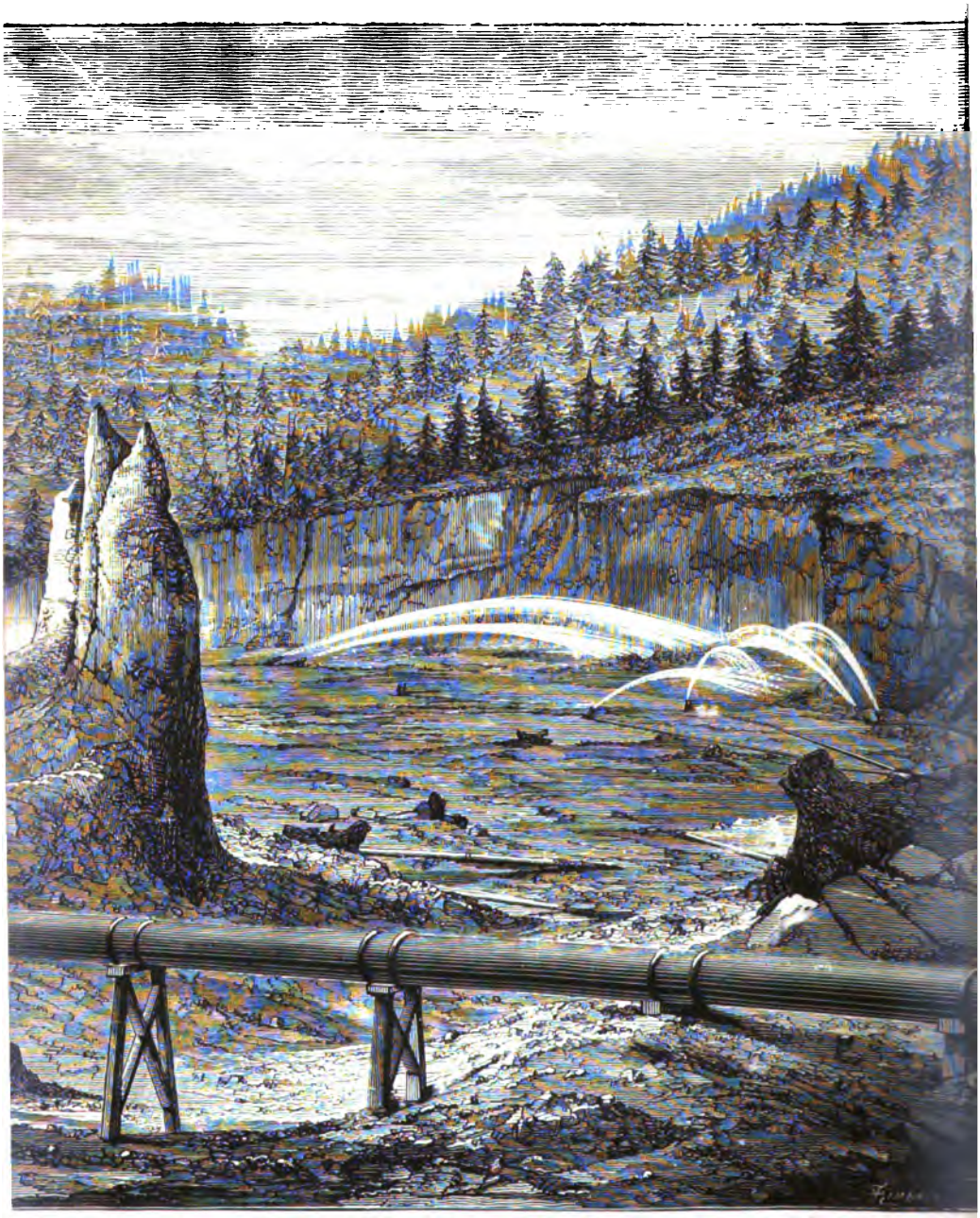
turns on friction wheels which are shown at E, while the vertical one is on pivots. A counterpoise I keeps the machine in position. The movement in all planes is given by the "deflector" A D, which is turned by the handle F. It is a very simple machine, has no bolt in it, to catch what is carried into the pipe by the water, and is easily repaired.

The end of all these machines is made of cast iron or of brass carefully turned in a lathe, and is called the nozzle. It formerly varied from 4 in. to 5 in. in diameter according to the power it is required to use. The tendency of the last ten years has steadily been to increase the power and size of the giants, so that the diameter of the nozzle has been gradually enlarged until it has now reached 11 in., and it is likely that in the near future they will be made even larger than that.

In order to prevent the rotary motion of the water, which is very apt to take place when it is discharged at high velocities from long tubes, the nozzle is provided in the inside at the supply end of the pipe C, with three rifle plates, which force the water to issue in a straight line from the nozzle, and thus prevent the breaking up of the stream, whose effectiveness depends upon the round shape being preserved.

The deflector was invented by H. C. Perkins, of North Bloomfield, in 1876, to produce automatically a change in the direction of the stream. It consists of a short piece of pipe A, Figs. 23 and 24, with a ball-and-socket joint which is screwed on to the butt end of the pipe C. The nozzle fits also the end of the butt, and may be screwed to it without the deflector if desired. When the lever F is moved in any direction it turns the deflector. The current of water passing through the pipe impinging against the deflector causes the giant to turn in any direction and in any plane into which the deflector is turned. As the whole weight of the nozzle is counterpoised, the slightest movement of the deflector will cause the stream to turn in the direction wished. The use of the deflector does not cause the water to spray or in any way injure the stream. Before its use, when the water was at a pressure of 300 ft. to 500 ft., it sometimes required all the force of three men to move the giants, but with the deflector a single man with one hand can direct the stream at will. If it is desirable to throw a butt stream, the nozzle may be removed, but the deflector still controls the pipe. The ease of movement is of the greatest advantage, for, when it was difficult, the men, to save themselves trouble, were apt to let the pipe remain in a less advantageous position than it should have been, thus causing great loss both of water and of time. This need never be the case with this machine.

It has been objected to the deflector that it fell when the water was turned off the nozzle and there was no longer any pressure to keep it straight, and that there was always some danger attending standing beside it in order to direct the stream to any given point. In Fisher's form of giant the machine is held in position by ratchet and pawl. The whole pipe is thus operated by a single long lever, so that the pipe man can control both the angle and the direction of the stream, so as to use it with the least difficulty at will. Each giant is furnished with two nozzles. The machine, when it is set in place, is screwed to a beam from a projection cast on to the lower part. This is securely bolted to the bed-piece, both ends of which have holes made in it for the purpose of driving iron bars through them to secure the giant firmly. Other bracing will also have to be made, and in doing it it is always better to stake against the bed-piece



Figs. 25 and 26. Vie

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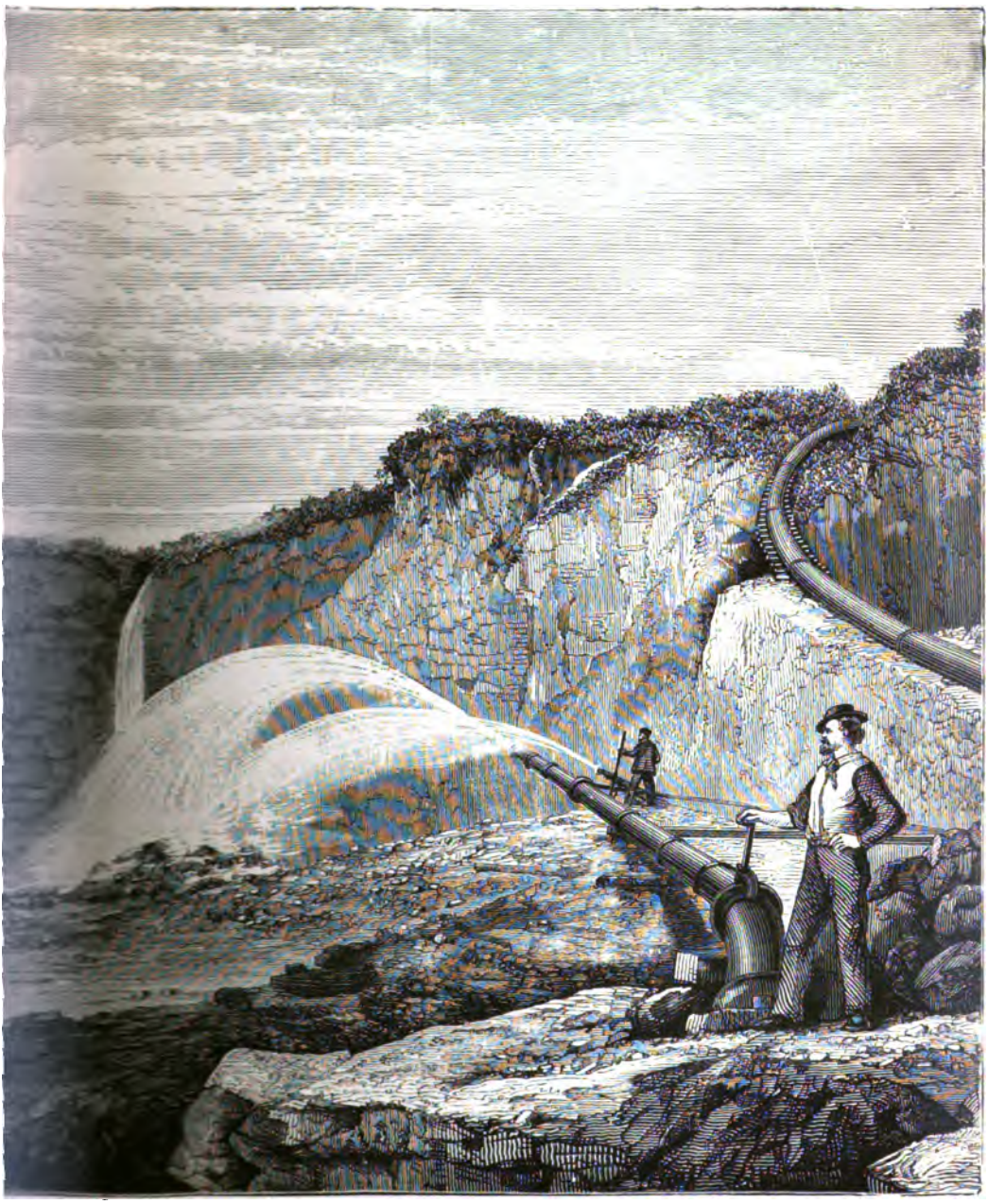


ELLIOTT BIGGINGS.

and brace against the stakes rather than to brace against the bed-piece directly. The nut of the main bolt, and the joint and the ball of the deflector, which can be easily reached through the nozzle, should be kept well greased. Tallow is always preferable to oil for this purpose, and when the water is very cold axle-grease can be used to advantage, as it does not break up and crumble like tallow.

The Table below gives the sizes in inches and weights of the double and single-jointed giants.* When larger sizes are required, they are sometimes made with rollers in the lower joint.

Nos.	Inlet.	Outlet.	Diameter Nozzle Butt Inside.	Weight of Double-jointed Giants.	Weight of Single-jointed Giants.
	in.	in.	in.	lb.	lb.
1	7	5½	4	275	200
2	9	7	5	450	360
3	11	7½	5 or 6	720	610
4	11	9½	7	960	675
5	13	9½	8	1200	770
6	15	9½	9	1550	
7	15	11	10	1860	

The number of nozzles which are to be used in any one claim varies very greatly. By far the largest number of companies have only one. Many have two, some have three, a few from four to five. At the La Grange Mine in Stanislaus County, California, there are twelve. At the Ben Wilson Claim, Boisé County, Idaho, there are thirteen. At the Spring Valley Mine, Cherokee Flat, Butte County, California, there are fourteen.

Fig. 25 shows a distant view of six nozzles at the Malakoff diggings, California, playing on a single bank. Fig. 26 is a near view of two of the same nozzles. Generally each nozzle has its own line of pipe, but occasionally as many as three nozzles are supplied by the same feed pipe. The best method of supplying the nozzles when only two are used is to fork the supply pipe. If more nozzles are required the branches are forked, and so on until the requisite number is reached. This method is the one which checks the flow of the water the least†. The size of the

* As made by J. Hendy, of San Francisco, and R. Hoskins, of Marysville, Cal.

† Letter of Mr. R. Hoskins, Jan. 8th, 1888.

nozzle varies from 1¼ in. in the smaller mines up to 11 in. in the largest. By far the greatest number are below 6 in. Those of from 1 in. to 4 in. throw streams of 30 ft. to 100 ft. long. Streams of from 100 ft. to 150 ft., and even in exceptional cases as high as 275 ft. long, are used. The length of the stream and its velocity depend, of course, upon the head. To be effective the stream must be solid when it strikes the bank. The water is frequently discharged from these nozzles with a velocity of from 75 ft. to 150 ft. per second, and will strike an ordinary bank from a distance of 150 ft. to 200 ft. with such force as to undermine it. A discharge of from 1000 in. to 1500 in. of water, which is equal to from 1500 to 1760 cubic feet per minute under a pressure of 150 ft. to 500 ft., is often used. The power of the water, which weighs often as much as 50 tons, projected at such an enormous velocity, seems to be absolutely irresistible, yet it is sometimes insufficient without the aid of powder to break down the bank. The velocity of the discharge is so great that a man striking the stream near the nozzle with an iron bar will have it twisted out of his hands without affecting the shape of the current of water at all, and the pressure is so great that the water will sometimes retain its cylindrical form unbroken for more than 100 ft.

To give an idea of the immense power which may be derived from water under pressure, it may be stated that a quantity of water equal to a thousand miners' inches can be discharged under a pressure of, say, 300 ft., through a 6-in. nozzle, with a velocity of 140 ft. per second, and a weight of 1650 lb. during the same period of time. Such a volume, uninterruptedly impinging upon a bank of earth or gravel, having, as it does, one-tenth of the velocity of a cannon ball, must necessarily do great execution, and produces the caving of an ordinary gravel bank without the necessity of resorting to blasting. Rocks weighing a ton and over are moved by the force of the water, and when in the way are sometimes rolled by the stream for short distances out of the road. It would not be expedient to use the water for such purposes except occasionally. Rocks, when they are not immediately in the way, are more economically moved by derricks.

The following Table gives the flow of water through nozzles* of various sizes :

Head.	Diameter of Nozzles.						
	2 in.	3 in.	4 in.	5 in.	6 in.	7 in.	8 in.
ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.
50	1.15	2.59	4.60	7.19	10.36	14.10	18.40
100	1.63	3.66	6.52	10.17	14.64	19.94	26.08
150	2.00	4.48	8.00	12.46	17.92	24.42	32.00
200	2.30	5.10	9.20	14.34	20.64	28.20	36.80
250	2.58	5.80	10.32	16.09	23.20	31.54	41.28
300	2.82	6.30	11.28	17.62	25.44	34.54	45.12
350	3.05	6.84	12.20	19.04	27.30	37.32	48.80
400	3.26	7.32	13.04	20.35	29.28	39.89	52.10
450	3.46	7.76	13.84	21.59	31.04	42.31	55.36
500	3.64	8.20	14.56	22.75	32.80	44.00	58.24

The total amount of water which will be required to wash a given quantity of gravel is exceedingly variable, and may range from single to more than double, varying with the situation and compactness of the bank, and the size and grade of the sluices. In low grade sluices it may be as low as 1.5 cubic yards per miners' inch ; in high grade sluices as much as 5 cubic yards per inch are moved. The size of the nozzle and the pressure of water will therefore be regulated in each particular case according to the circumstances. It will generally be found best however to arrange for a maximum supply and pressure. If it is necessary to make any changes in the supply it can be regulated from the distributor. It is generally calculated that with a fall of $\frac{1}{4}$ in. to the foot it will require 7 cubic feet of water to wash 1 cubic foot of gravel.

Almost every company has more water than it can use, which they therefore sell to increase their profits. Many of the companies have made exact measurements of the quantity of water, which is all the more necessary since the water is sold to the mining companies. The water is paid from 10 to 20 cents the inch per day, these days varying from ten to twelve hours. It has frequently happened that the working of gold, which at first was the principal object of these water companies, has become only a part of their profit. They work the gold and sell the

* Prod. Gold and Silver in the U.S., 1884, p. 572.

water at the same time. It is evident that such undertakings as furnishing the water for very large mining enterprises cannot be done except by great accumulation of capital.

Cost of Construction.

The following Table* gives the expenditures in the construction of reservoirs, ditches, and bedrock tunnels in operating hydraulic mines in California:

	$
North Bloomfield and Milton	4,079,321
Excelsior, of Smartsville	2,857,166
Gold Run	433,335
Blue Tent	755,000
El Dorado, Deep Gravel	1,008,611
Eureka Lake	2,500,000
South Yuba	2,000,000
Spring Valley...	2,000,000
La Grange	650,000

The following Table† gives an idea of the magnitude of this interest in the United States:

Total length of ditch lines	10,873 miles
,, ,, ditches proper	10,183 ,,
,, ,, flumes	450 ,,
,, ,, pipes	150 ,,
Greatest length of lines under one ownership	290 ,,
Average length of ditch lines reported, including feeders, &c.	26 ,,
Average length of all California lines	12½ ,,
Maximum capacity of ditch lines (per twenty-four hours)	7,560,000,000 gallons
Total cost of plant, including reservoirs, &c., and excluding cost of water rights	$ 27,056,942.11
Average cost per mile, including reservoirs, &c., of lines reported	3,800.60
Average cost per mile, including reservoirs, &c., of all lines	2,509.22
Highest cost of plant reported under one ownership	2,100,000.00
Nominal capital stock of thirteen corporations, exclusive of mining companies owning ditches	38,805,000.00
Working capital of twenty-seven lines... ...	1,451,300.00
Total annual expense of maintenance... ...	827,280.41
Annual cost of repairs	200,288.66
,, ,, attendance, exclusive of repairs	626,991.75
Average length of season of lines reported ...	211½ days

* Prod. Gold and Silver in the U.S., 1883, p. 708.

† Report of U.S. Census, 1880, vol. xii., p. 205.

What the amount actually invested in these water companies is can only be estimated. The amount of $38,805,000 given in the preceding Table is nominal capital which bears no assignable relation to the actual cost or value of the plant. What the amount of working capital is which is required to run a hydraulic plant is estimated as between $200,000 and $750,000. The differences are very wide and depend entirely upon the size of the plant. That it need not be at any time much larger is owing to the fact that the returns are large and quickly handled, and the gold always finds a ready sale. The total amount of working capital depends too on the extent of ground opened and being opened while the work is progressing. The enterprise must be a very large one which would require more than the larger amount of the two limits. The amount of invested capital seems to be about $27,000,000 exclusive of the purchase of water rights. In the estimate is included ditch lines, flumes, and pipes. There are in the United States nearly 11,000 miles of these ditch lines. The following Table* gives the cost of the construction of a number of miles of these lines in different States of the United States:

State or Territory.	Number of Ditch Lines reporting Total Cost.	Corresponding Number of Miles.	Average Cost per Mile, including Flumes, Pipes, Reservoirs, &c.	Cost.
			$	$
Alabama	1	1	586.00	586.00
Arizona	1	16	3750.00*	60,000.00
California	60	1667½	4698.97	7,835,539.26
Colorado	5	30¼	2925.62	88,500.00
Dakota	6	61¾	9379.76†	579,200.00
Georgia	5	92¼	5767.97‡	532,095.00
Idaho	16	286	1296.28	370,735.70
Montana	9	105	2738.10	287,500.00
Oregon	51	576¾	1779.38	1,026,258.35
Total	154	2836½	$3800.60	$10,780,414.31

* Pipe line. † Including an expensive flume line. ‡ Including a large proportion of pipe.

It is interesting to note by the above Table that while these ditches are for the most part confined to the States of California and Oregon, they are increasing in use in the east, and are coming

* Report of U.S. Census for 1880, p. 214.

into use in very many States which were not reported to the census for uses in the extraction of other metals than gold and silver. In some of the Middle and Southern States, stripping for iron mining is extensively done by water.

It appears that 154 ditch lines, with a total length of 2836½ miles, cost $10,780,414.31, or an average of $3800.60 per mile for all plant, including flumes, pipes, reservoirs, buildings, &c. The 213 miles of ditch lines reported to the census, without particulars as to cost, are assumed to have cost quite as much per mile as the 2836½ miles for which the expense is specified, or $809,527.80, bringing the total cost of 3049½ miles reported on to $11,589,942.11. The average cost per mile of the lines not included in the reports cannot, however, be nearly so great as that of the ones which, because of special engineering difficulties, extent, or capacity, were chosen for minute investigation; for although some important works unfortunately escaped the canvass of the census agents, the bulk of the lines omitted were of less costly construction than those included in the returns. Assuming an average cost of $2000 per mile for these lines, the whole expense of building 10,783 miles (the estimated total length of ditch lines in operation) would be:

	$
Cost of 2836½ miles, at $3800.60 per mile, as per schedules	10,780,414.31*
Estimated cost of 213 miles for which the schedules do not state cost, at $3800.60 per mile	809,527.80
Estimated cost of 7733½ miles, at $2000 per mile ...	15,467,000.00
Total cost of 10,783 miles...	$27,056,942.11

This is exclusive of the purchase cost of water rights, for which the aggregate amount paid is not ascertainable.

The length of the season during which the water is available is very variable. When there is large storage capacity it can be used the whole year round, both day and night. When there is not, the number of days in the year and even the number of hours per day is limited. This often enables the small reservoirs to be filled in the night when their supply is drawn down by use during the day. It appears from the following Table† that 142 water lines give a general average of 211½ days, with a range between an average of 135½ days in Montana and a

* Report of U.S. Census, 1880, vol. xiii., p. 214. † *Ibid.*, p. 219.

continuous supply in Georgia. Regarding these results, two essential qualifications should be noted: First, that the averages are computed solely from the limited schedule data; and second, that as the investigation included mainly the more important lines, the probabilities are that considerably higher averages are obtained than would be the case were all the lesser lines taken into account.

Average Length of Water Season.

State or Territory.	Number of Ditch Lines reporting Length of Season.		Average Number of Days.
Alabama	1		365
Arizona	3		288⅓
California	48		242¾
Colorado	8		153⅙
Dakota	6		264⅔
Georgia	8		365
Idaho	17		153½
Montana	7		135½
Oregon	44		178½
Total	142	Average	211½

The following Table* gives the amounts of water used in the Sacramento and San Joaquin basins in California, expressed in twenty-four hour inches.

Quantity of Water used in Mining in the Sacramento and San Joaquin Basins in 1880.

Name of Stream.	Twenty-four Hour Miners' Inches.
Table Mountain, or Dry Creek	833,250
Butte Creek	24,000
Feather River	1,259,363
Yuba River	5,458,171
Bear River	1,117,082
Dry Creek No. 2	44,229
American River	1,914,500
Cosumnes River	504,000
Dry Creek	162,000
Mule Creek	38,000
Sutter Creek...	457,750
Jackson Creek	171,150
Mokelumne River	224,500
Calaveras River	975,750
Stanislaus River	38,720
Tuolumne River	812,500
Total	14,034,965

* *Ibid.*

Taking the miners' inch at 2250 cubic feet per twenty-four hours, the total quantity used in this region represents a sheet of water 37¾ miles long, 3 miles wide, and 10 ft. deep.

Such enormous expenditures of money, such commercial enterprise, and such daring energy have been displayed in the construction of these waterways and the application of the power derived from them as is almost without a parallel in the history of hydraulic engineering. It has had the effect to produce not only great engineering works but great engineers. It has been considered by some a matter of great misfortune that the hardy, adventurous, and self-reliant miner has totally disappeared from California to be replaced by corporations without souls and men who work for day's wages; but this is short-sighted. The old-fashioned miner has gone, it is true, but the skilled engineer has taken his place, and the engineering and commercial world has been greatly benefitted by the change.

HYDRAULIC MINING PROPER.

I. Construction of the Tunnel and its Connections.

To put the ditch lines to use the ground containing the gold must be carefully surveyed to ascertain the exact position of the rimrock, the length of the bed, and the direction of the channel. A careful topographical survey of the whole country included within the limits of the claim must therefore be made. Its peculiarities and capabilities having been ascertained and carefully discussed, a cut or tunnel or both must be run so as to reach not only on the one hand the lowest level of the channel in the claim, but also on the other to give the lowest practicable outlet for the deposit of the tails.

The great majority of the hydraulic mines are opened by cuts, and comparatively a small number are opened by tunnels alone, and some by both tunnels and cuts. They are always a great source of expense in opening the claim or bottoming the bedrock, as it is called. When the cut has to be made in rock it is blasted out. When a tunnel is to be driven it is done exactly as it is done in ordinary mining, except that, as the tunnel is to

be a permanent structure through which large amounts of water and gravel are to be run, it must be timbered with the very greatest care. When the cut is in earth, if it is loose, it is made with a hydraulic nozzle, which is the cheapest method of excavation, or when it is hard, by the ordinary methods of digging. Occasionally the gravel is so hard that it is desirable to use power drills driven by compressed air. They are always used in tunnelling, and their use in the gravel sometimes greatly facilitates the work. The gravel itself makes no difficulty except so far as timbering is concerned. The explosives which are generally used in blasting the cuts and tunnels are Giant, Hercules, or Vulcan powder, which are always fired by time fuzes.

The tunnel may be called the main artery of a hydraulic claim. If the gravel contains the pay, the success of the claim will depend, other things being equal, on its proper location and construction. It should be perfectly straight, free from curves, not only because this is the simplest way to run it, but because it will be the shortest, and therefore will involve the least expense, and because if a circuitous tunnel were attempted the miners would be liable to lose themselves altogether. The length of the bedrock tunnels, as they are called, varies considerably according to the kind of ground, and also the extent of the claim which it is intended to open. It usually varies from 500 ft. to 600 ft. to 3000 ft. or 4000 ft. In one instance, that of the North Bloomfield tunnel, the length is 8000 ft.

It is generally best so to arrange the grade of the tunnel, that it will remain in the bedrock at the lowest point of the channel, the reason why it is made to come out below being that the surface of the bedrock being very uncertain the irregularities in it cannot be determined beforehand. It is therefore necessary to provide against these irregularities being found before it is too late, and to bring the bedrock tunnel below any possible depression in it. The preliminary survey having ascertained, either by shafting or by drilling, what the exact depth of the channel is, the tunnel must be so run as to reach its lowest point, and thus command the whole of the ground, so that when the bedrock is known to be irregular the tunnel is sometimes

made to end as much as 30 ft. or 40 ft. below its surface, and is connected with it by means of a shaft which is called a chimney or an incline. It may, however, happen that quicksands or other difficulties make this plan impracticable, so that the lowest point which can be reached under the circumstances must be selected and this point connected with the tunnel by a shaft or incline. With the modern methods of passing quicksands they do not now present any difficulties which may not be overcome without any very great expense, and they are no longer the terror of the miner, that they formerly were, when, owing to the great pressure under which they existed on the side of the rimrock, they would frequently break through with but very little warning, and thousands of carloads of sand and water mixed would burst into the tunnel, all of which had to be removed with the greatest care and precaution in order to avoid a subsequent repetition of the same thing before the work could progress any further, thus retarding the progress of the work and increasing the cost. It is said that one of the inclines of one of the greatest hydraulic mines took four years to construct owing to a succession of accidents of this kind, and cost not far from $100,000, when under ordinary circumstances it would have cost less than a third of that sum.

In determining the exact location of the tunnel it must be run in such a way that from its mouth on, there would be sufficient fall for all the sluices, undercurrents, and other contrivances to catch the gold, and still have velocity enough at the end of them all to deposit the tails on the dump. It is best generally not to make the connection with the tunnel from the rimrock on account of the large bodies of quicksand and of loose gravel which usually lie upon it, and which may very much lengthen and complicate the work. In the year 1856 a tunnel was commenced at Sucker Flat, on the Yuba River, which was not completed until, after eight years of great labour, in the year 1864. The claim was rich, and large fortunes were made from it, but after working several years it was found that from defects in the survey the tunnel did not reach the bottom of the bedrock by some 30 ft. To get at this, which was the richest part of the claim, another tunnel 1500 ft. long had to be constructed. At the Pioneer Mine, Grass Flat,

Plumas County, the claim was opened by a tunnel 4000 ft. long at a cost of $60,000. When midway in the channel the tunnel was found to be 22 ft. above the bedrock. The whole work was a total loss. The tunnel had to be commenced anew, and $100,000 more was spent before the mine was opened properly.

The tunnels of the Gold Run Ditch Company are over 4000 ft. long, and are made 8 ft. by 10 ft. Those of the Cedar Creek Company, at Dutch Flat, are nearly 4000 ft. long. At the Nevada Mine, near You Bet, in Nevada County, the tunnel is nearly 3000 ft. long. Near Smartsville there are several tunnels which are from 2000 ft. to 3000 ft. in length. Most of these tunnels were run with a diamond drill. Except when the bedrock is so soft as to be liable to wear into holes, the tunnel itself may be used as a sluice. When such is the case wooden sluices are laid through their entire length. 1800 ft. of the North Bloomfield tunnel have been paved with blocks, the rest, as it was a tough slate with upturned edges which answered for riffles, was left bare. The cost of the tunnels varies with the kind of rock, their length and size, and varies from $20 to $60 the foot.

The preliminary survey for locating a tunnel should include a reconnaissance of the surrounding country, so as to find the lowest outlet of the region. In selecting this point the grade of the tunnel should be fixed at 3 ft. to 4 ft. 6 in. to the 100, and the depth of the starting point should be so arranged that with this grade it will come out in the claim below the rimrock, on the opposite side from which it enters the basin. Figs. 11 and 27 show a N. and S. section of one of the large mines at Sucker Flat,

FIG. 27. SECTION AT SUCKER FLAT.

near Smartsville, on the Yuba River, California. It shows the condition of the mine in August, 1874, when this section was made. *a* is the bedrock in which the Yuba River has cut its channel *g*. The bottom is filled to a certain height by the tails, as shown at *b*. Above this other tails *d* are deposited, within

which are the present river banks. Two tunnels are shown in the gravel, one at *f* at a high level, which, run above the bottom of the rimrock, allowed of working only a small part of the deposit, and another, *e*, lower down, from which the whole of the ground was then being worked. The upper one was very short, and was driven to get ground to pay while the longer one was being constructed. With it the ground between the hills *h* 1, 2, and 3, and the rimrock V on the one side and W on the other, were worked out, and when the lower one was completed the whole of the rest of the ground was made available. The ground was worked out by the upper tunnel, leaving rather high hills, numbered 1, 2, 3, as witnesses to what the ground formerly was. From the top of the hill No. 2 to the mouth of the lower tunnel was 300 ft. The ground was worked in terraces at three levels. The tails were all washed into the Yuba River. It is not generally considered expedient to work with more than one tunnel, and that one run as low as possible, but in this case it was good policy to run the upper tunnel *f*, as the money to carry on all the works was made out of the claim with the small expenditure of the first tunnel as invested capital.

The Bloomfield Mine, situated in Nevada County, California, in what is known as the San Juan Ridge, runs its tailings into the South Fork of the Yuba River. It is an elevation of 3200 ft. above the sea level. The company owns a large tract in the ancient river channel. To explore this ground and ascertain the exact position and depth of the ancient channels four shafts were sunk to the bedrock, only one of which struck the main channel. This shaft showed the thickness of the gravel to be 207 ft., of which the lower 135 ft. was the blue gravel which characterises the ancient river deposits. From the bottom of these shafts 2000 ft. of drifts were run in the direction of the channel, which proved to be about 500 ft. in width. The 21,614 tons of material extracted from these drifts yielded $36,600 in gold at a cost of $63,956. The tunnel was located on the data obtained from the survey. To carry on the work with sufficient rapidity, eight double compartment shafts from 800 ft. to 900 ft. apart, and shown on the map, Fig. 28, were sunk. The shafts were 4 ft. 6 in. by 9 ft. in the clear. The hoisting and pumping were

done in the different compartments by hurdy-gurdy wheels 18 ft. to 21 ft. in diameter. The water to move them was delivered under a pressure varying from 240 ft. to 500 ft. The length of the tunnel is 7874 ft. For 6000 ft. from its mouth the tunnel is 6 ft. 6 in. high and 6 ft. wide. For the rest of the distance it is 8 ft. by 8 ft. The progress made in driving it was something more than 100 ft. per week. As the ground was wet about 500,000 gallons of water had to be pumped every twenty-four hours. The construction of the tunnel lasted two years, keeping 160 men constantly employed. The total cost was over $550,000.

The method of using power in driving the North Bloomfield Tunnel* is an excellent illustration of the advantage of the use of the hurdy-gurdy wheel. It was originally about 8000 ft. long, and was run through a very highly metamorphosed slate with its general line passing under a good-sized stream at a depth of about 190 ft. The eight working shafts were about 200 ft. deep, which, with the lower entrance, gave sixteen working faces. Diamond drills were used at the lower heading, requiring power. The other fifteen headings were driven by handwork. It was uncertain how much water would be encountered, but from the location it was evident that a large quantity might be struck at any shaft, and hence it became necessary to have ample power at hand at each opening in readiness for such an emergency. A pipe main was laid along the general line of the tunnel with its penstock 285 ft. vertical above the surface at the upper shaft, and 549 ft. from the lowest. It was made of single-rivetted sheet iron, No. 14 of Birmingham gauge, in lengths of 20 ft., put together stove pipe fashion, with the joints made tight by tarred cloth strips and pine wedges. This pipe was 15 in. in diameter at the penstock, diminishing to 13 in. and 11 in., and was 7 in. at its lower end. From it short branches 7 in. in diameter were extended to the several shafts. It was in one case carried across the stream by a light suspension bridge 150 ft. long, the trunk of a tree on either side forming the towers for supporting it. The aggregate length of the main and the branches was 9960 ft., with some

* Trans. Am. Soc. Civil Eng., vol. xiii., p. 23.

* Mining Commissioners' Report, 1875, p. 118. † 546.5 ft. hand work, 630.9 diamond drill work. ‡ Diamond [illegible]

Résumé of the Cost, including Water and Air Pipes, Tracks, Cars, &c., and Diamond Drills.

Total cost of shafts		$101,478.76
,, ,, headings		274,892.37
Foundry work:		
Cost in San Francisco and Nevada City	$30,597.96	
Freight to Bloomfield	1,723.09	
		32,321.05
Erection, track, &c.:		
Timbers and lumber	7,389.71	
Track and cars	3,397.26	
Labour	18,958.46	
Hardware, nails, &c.	4,149.05	
		33,894.48
Water power pipes, gates, &c.:		
Cost in San Francisco	12,113.62	
Freight	793.57	
Labour and lumber in laying	1,724.25	
		14,631.44
Drill:		
Cost of drill, pipe, and fixtures	3,373.40	
Labour in fitting	1,875.43	
Lumber	327.36	
Supplies	61.50	
		5,637.69
Diamonds, purchased 238 carats ...	1,914.96	
Less diminution in weight and value by use	700.00	
		1,214.96
Carried forward		$464,070.75

General Expen[illegible]

Brought forward		
General work and roads:		
Foreman, machinist, pump men, track layers, general smithing, fitting up pumps, &c.	$19,689.02	
Roads	2,122.76	
Buildings and tools for general use:		
Feed for teams, firewood, and general supplies	5,590.22	
Surveys and administration:		
Salary of company's supt. and accountant (part), and assistants, taxes, &c....	17,086.38	
		44,488.38
		508,559.13
Deductions.		
Materials sold and transferred:		
Pipe and pumps sold	1,037.27	
Water pipe and gates (part), T iron, cars, set of machinery at No. 7 ...	8,695.70	
		9,732.97
		$498,826.16

[*To face page 203.*

hurdy-gurdy wheel set on the rear of the drill carriage. This was afterwards modified by placing a separate hurdy-gurdy wheel on a sleeve on each drill rod, the advance movement of the drill being given by hydrostatic pressure on an annular piston, thus doing away with all gearing. These eight sets of machinery were run for 2½ years, the only break being that of a spurwheel, caused by the careless dropping of a steel bar between it and its pinion. Aside from this accident, not a dollar was spent for repairs, and the machinery, including the pipe, was in about as good order when the tunnel was finished as when it was first erected. One man on a twelve-hour shift operated the machinery at each shaft beside dumping the cars. Two men kept the eighteen pumps on the line clear, the principal work being keeping the suction pipes for the down grade headings tight. A force of eighteen men only was required for the eight shafts. The cost of pipe, gates, &c., when put in place, was $14,631, and the cost of machinery was $60,000.* The detail of the cost of this tunnel is given in the annexed Table.

The mine is being worked at both ends;† that is, it is following the old river channel both up stream and down stream, delivering its tailings into the bedrock tunnel through two branch tunnels, one at the lower end of the mine, 871 ft. in length, and the other at the upper end, 1400 ft. long, Fig. 28, known respectively as No. 8 and No. 10, so named from the numbers of the shafts at the ends of the branches. Leading to these branches from the headings of the mine are, at the down-stream end, a sluice 550 ft. long, paved with block riffles, and an open rock cut 700 ft. long; and at the up-stream end an open rock cut of 2000 ft., which is run in the same grade as the sluice in the tunnel, viz., 4 ft. 6 in. to the 100 ft. The sluices in the branch tunnels are 5 ft. 9 in. in the clear and 30 in. deep, the bottoms being paved with block riffles 15½ in. by 15½ in. by 13 in. deep, and the sides protected with side lining blocks 18 in. by 18 in. by 3 in. thick, which reach from the top to within 3 in. of the bottom. Below the branch tunnels which come together at shaft No. 7 comes the rock tunnel, 7000 ft. long, at the mouth of which there is a system of tail cuts,

* *Ibid.* † School of Mines Quarterly, vol. v., p. 373.

sluices, and undercurrents, which all serve to catch the amalgam and free gold that have passed the other parts of the mine. The banks of gravel through which the gold is distributed are washed down and the material runs through the cuts, &c., into the sluices

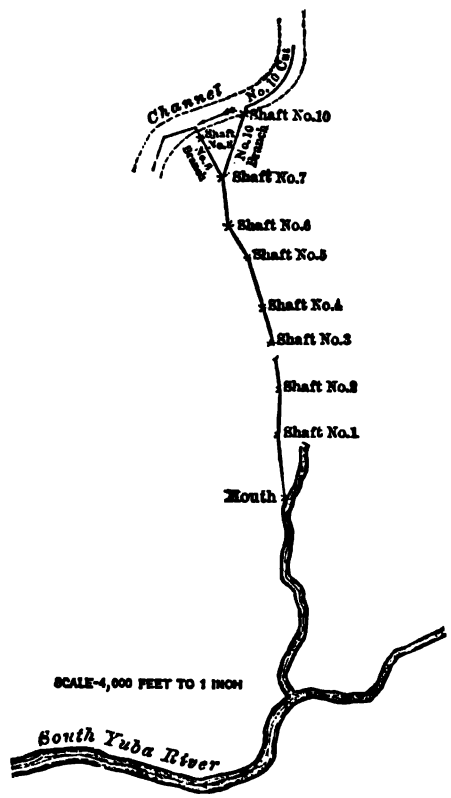


FIG. 28. PLAN OF THE NORTH BLOOMFIELD TUNNEL.

in the branch tunnels, where the larger part of the gold is caught by the quicksilver in the spaces between the riffle blocks. After passing through the sluice, the material, with what gold is not caught, runs through the rock tunnel, tail cuts, tail sluice, and finally through the undercurrents, after which it finds its way into the Yuba River.

The size of a tunnel will depend upon the dimensions of the sluice which is to go through it, and it may be said in very general terms that it should be from 2 ft. to 2½ ft. larger than the flume, so as to give sufficient room to the workmen, when cleaning up, and to give plenty of space for piling the paving blocks or

stones when they are taken out. It will also depend upon the quantity of water and material that is required to pass through it, and the length of the mining season. They are now generally constructed 7 ft. to 12 ft. high and 6 ft. to 10 ft. wide for single sluice boxes, where the mines are supposed to run eight or nine months in the year. A tunnel of such a size, with a flume 6 ft. square, and 4 per cent. grade, will use from 2000 in. to 2500 in. of water. Where a larger supply is needed, and as high as 3000 in. to 3500 in. of water can be had, a larger size and a double set of sluices would be desirable. Tunnels of 5 ft. and 6 ft. were formerly considered to be great works, but of late years, since the introduction of drilling machines, tunnels of double that size are often undertaken without hesitation.

The general grade will be 4 per cent., or 6 in. in 12 ft. The larger the sluice the less the grade, for a heavy grade in a large sluice would be likely not only to carry off the sand, but perhaps even to deteriorate the sides and pavement of the sluice. It is desirable to have as rapid a grade as is consistent with the greatest yield in gold, for this is one of the best means of disintegrating the sand and liberating the gold to the action of the mercury. A very coarse and compact gravel may require a grade of 5 to 7 per cent. Where the survey shows a 4 per cent. grade to be very difficult to attain and the gravel is light, 3½ per cent. is sometimes used. It is generally an expensive matter to use a low grade, since what is not carried off by the water will have to be moved by hand. A remedy for a too steep grade can always be found, but a too low grade has no remedy without great outlay, frequently equivalent to an entire reconstruction of the tunnel, and will always be a source of expense. The lower the grade the less the amount of water required, and the higher it is the greater this will be. It has been estimated that a sluice with a grade of 12 in. in 12 ft. will carry four times as much as one with half that grade. With a 12-in. grade boulders weighing from 200 lb. to 500 lb. pass easily and rapidly through the sluice, and those weighing from 1000 lb. to 2000 lb. will pass slowly. The rule to be followed with regard to the tunnels is, that they should be carried as far as practicable in such a way that the lowest point on the bedrock

can be worked, and the main channel should therefore be the shortest line of outlet from this point. In any case the outlet of the tunnel must be from 50 ft. to 60 ft. below the lowest point of the bedrock, Figs. 11 and 27. The exact depth will generally be determined by the more or less firm nature of the rock. The lowest point possible should be sought, every inch that can be gained consistent with the proper working should be taken advantage of. It is generally difficult, however, to gain more than 60 ft. or 70 ft., owing to local causes. In running the tunnels special provision should be made for ventilation in case of obstruction, for it often happens that the shaft or the tunnel becomes clogged, and when the tunnels are very long, as was the case with some of those which became clogged about Smartsville, they may become very foul. One of these, where no precaution of this kind was taken, was 1800 ft. long; the air became so foul while it was being cleaned, that it was impossible for the workmen to remain any length of time in it. It is always judicious either to provide an air shaft or air boxes to ventilate in case of accident; the first cost of the air shaft is greatest, but it is a wise precaution; in many cases it would be well to provide both.

The tunnel should be run to as near the centre of the basin as practicable before it is connected with the surface workings, and should on no account be opened on its edge or near the rimrock, both because quicksands and poor pay are likely to be found there, and because the expense and difficulties of opening are about as great near the edge as at the centre. The mining ground opened at the centre will last for a long time, while that opened at the edge will only last for a short time. The ground should be lowered in every direction from the shaft very evenly before working is commenced, or any attempt is made to take off the top timbers of the shaft. If the ground is not even, the shaft, and sometimes the tunnel, will become clogged by the rush of material from the lower sides only, which will often cause great delay, and sometimes even danger to life, before they can again be opened. There are three general forms which are given to the shafts. They may be vertical when it is an ordinary simple shaft, they may be in-

clined like a shoot, or in the shape of a terrace. If the shaft is used it should be as near vertical as possible, so as to utilise the whole height of the fall and break up any compact gravel, and render it more easily attacked in the sluices. It should be constructed in such a way as to insure permanence. The shaft should therefore be well secured throughout its whole length. If it is designed to be permanent, it should be heavily and strongly timbered, and lined on the inside to within 8 ft. to 10 ft. of the surface with blocks 6 in. to 8 in. thick, in such a way that the wear will be upon the lining and not upon the timbers. This is especially true near the bottom, where most of the wear comes; and the lining at this point should be of very heavy timber. The connection with the tunnel should be made with very great care, for it has sometimes happened that from want of care in making it, quicksand has been entered, which has filled the tunnel in a few minutes, and necessitated the construction of a new one. Proper precaution would require the running of a drill some distance ahead and the testing of the material from the hole so formed. If on being tested the ground is found to be hard, the tunnel can be run with safety to the shaft. If it is soft, it must either be abandoned or else worked with great caution. Should it be found that the ground is soft in every direction in which it is desirable to prosecute the work, it will be better to make the connection between the shaft and the tunnel by means of a slope, rather than to continue the shaft vertical for its whole length. When the work becomes dangerous an arrangement must be made in some way to protect the end of the slope and leave the water sufficient time to drain without giving the sand an opportunity to move. When this has been done a careful survey of the whole ground must be made to ascertain the exact spot where the shaft can connect with the slope. If the tunnel has been made perfectly straight this matter presents no difficulty. The exact point having been ascertained, the shaft or chimney is sunk in the usual way, and well secured. If it drains itself there will be no difficulty in going through any sand or gravel that the shaft may have to traverse. If it does not free itself entirely from water, pumping may be resorted to; or if the ground is moderately

firm, a hole may be drilled from the top to the bottom of the shaft or incline, and a drainage outlet to the tunnel formed. Owing partly to irregularities in the bedrock, but chiefly to frequent occurrence of quicksands, the shaft of one of the mines took four years to sink, and cost about $75,000. This experience, however, has been profited by in every shaft sunk since then.

There does not seem to be any special advantage in an inclined chimney except that engineering difficulties may make it expedient to connect in this way as being the least expensive or the most expeditious solution of the problem. If such a chimney is very short it may be changed after a little into a shaft, or if it is long it may be worked into a terrace shape, which will have the advantage of breaking up, by the fall, any of the gravel which is held together by a very hard cement. These terraces should be arranged so that the fall at the top will be the greatest, and the other ones less. In a fall of 100 ft. the first terrace should be 30 ft., the second 25 ft., the third 20 ft., the fourth 15 ft., the fifth 7½ ft., and the last, where the material drops into the sluice, 2½ ft. It will be seen at once that a vertical shaft is nothing more than a terrace with a single drop. It is a matter of discussion among engineers whether a single drop or a succession of drops are better. It would seem at first sight that a succession of drops would be more likely to break the material fine and expose the free gold than a single drop, but engineers are not agreed upon the subject.

If the tunnel already provided with a shaft must be pushed towards the opposite rimrock, or if the shaft is connected with the tunnel by an incline, and it is desired to continue the tunnel so as to meet a vertical shaft, a compartment which is water-tight and well-ventilated, and also perfectly safe, must be made in it, through which the miners may descend to prosecute the tunnel. It would not be safe, of course, to have that side of it connected with the sluice boxes, for a rush of water or a stoppage might overwhelm the miners running the tunnel towards the opposite side of the basin. A compartment 4 ft. by 4 ft. will be of sufficient size for the water. It sometimes, though rarely, happens that the lay of the ground is such that the preliminary operations may be commenced before driving a tunnel; this is generally the

case if the thickness in the centre of the basin is 200 ft. to 300 ft. This may be done by arranging ditches so that the mine may commence to pay at once, while the permanent tunnel for future operations is being driven below.

The connection with the surface is thus made by a chimney, and after junction has once been made between the tunnel and the surface the mine will last for a long time. This connection once established, the improvements and additions are easily made. When all the ground controlled by this chimney has been worked out, it is sometimes possible to extend the tunnel in such a way as to open up new ground, but usually when it is possible the chimney should be sunk in such a position that it will be capable of working out the whole of the ground. An idea of the expense of such tunnelling is given by the experience of the Gold Run Ditch and Mining Company, who built their tunnel 600 ft. below the crest of the ridge. They ran two Burley drills for twenty-four days during the month of May, 1864, and made 126½ ft. of tunnel through hard rock. The following Table gives the details of the expense :*

	$
XX Hercules powder	1064.43
Fuze and caps	15.00
Piping, air and water	129.25
Wood for boiler...	175.00
Coal	16.38
Steel and iron	50.00
Oil for lights and machinery	78.00
Lumber tracking	25.00
Labour	1400.00
	$2953.06

The Table† on next page gives the dimensions and grade of the tunnels of several of the principal mines:

* Min. Com. Rept., 1875, p. 101.

† Report of U.S. Census, 1880, vol. xiii., page 187.

CALIFORNIA.

Name.	Locality.	Length.	Width.	Height.	Average Grade.
		ft.	ft.	ft.	ft. per 100.
American* ...	Below San Juan	3900	...	...	6.25
Babb	Timbuctoo ...	1200	...	...	3.80
Bedrock	Below Sweetland	2600	...	...	5.25
Blue Gravel ...	Sucker Flat ...	1100	...	...	4.50
,, Point ...	,, ,, ...	2250	...	...	4.16
Bonanza	Mokelumne Hill	200	4	6	6.25
Boston†	Wolsey's Flat ...	1600	...	...	7.25
Buckeye	Buckeye district	1500	7	7½	
Deer Creek ...	Mooney's Flat ...	2200	...	...	3.40
Del Norte ...	Del Norte County	600	3	6	5.56
Dry Creek ...	Igo	300	...	...	2.78
Duryea	Mokelumne Hill	250	6	4½	5.56
English Mine ...	Badger Hill ...	1400	...	...	7.00
Enterprise ...	Sucker Flat ...	1200	...	...	4.16
Evans & Bartlett	Red Hill ...	300	...	...	4.17
Farrel	Columbia Hill ...	2200	...	...	3.50
French Corral‡ ...	French Corral ...	3500	...	...	4.66
Gold Run ...	Gold Run ...	2500	10	12	4.02
Hidden Treasure	Michigan Bluff...	2700	6½	5	1.32
La Grange ...	La Grange ...	*a*1300	6½	4	2.02
Lyon Gravel *b* ...	Placerville ...	1300	6	5	1.04
Mammoth ...	Red Hill ...	360	8	6	6.25
Manzanita§ ...	Sweetland ...	1740	...	...	4.16
North Bloomfield‖	Humbug Cañon	8000	...	...	4.50
Orion	Iowa Hill ...	600	8	6	5.56
Pactolus	Timbuctoo ...	1700	...	...	4.16
Paragon *b* ...	Bath	2000	5½	5 to 6	1.04
Penobscot ...	Mokelumne Hill	200	3	6½	4.86
Pittsburg ...	Sucker Flat ...	900	...	...	4.16
Polar Star ...	Dutch ,, ...	600	9	9	6.94
Rose's Bar ...	Timbuctoo ...	1600	...	...	4.16
Spring Valley ...	Cherokee Flat ...	1400	6	6½	2.01 to 4.02
,, ,, ...	,, ,, ...	400	6	6½	2.01 to 4.02
Sunnyside ...	Seneca	750	5	6¼	0.61
Sweetland Creek¶	Sweetland ...	2200	...	...	4.66
Van Emmons ...	Michigan Bluff...	*c*1800	4½	8	9.03
Weed's Point ...	Galena Hill ...	600	4 to 5½	6	4.51

* Cost $140,000. † Cost $140,000. ‡ Cost $165,000. § Cost $62,000.
‖ Cost $500,000, has eight shafts. ¶ Cost $90,000.
a Five. *b* Drift mines. *c* Two.

COLORADO.

L. H. Arthur ...	Grass Valley district	350	4	5	Level.
,, ,, ...	Grass Valley district	400	4	5	Level.
Rob Roy... ...	Spaulding district	50	5 to 6	4	0.42
,, ,, ...	,, ,,	67	5 to 6	4	0.42

DAKOTA.					
Castle Creek *b* ...	Cañon district ...	550	10	12	2.43
Fort Meade ...	Ruby ,, ...	625	10	12	2.43
Last Chance Bar *b*	Cañon ,, ...	1300	4	5	
McKay and Ross	Cape Horn district	200	5	7	4.02
GEORGIA.					
Findley	Lumpkin County	270	4	6	
Griscom	,, ,,	600	6½	6	8.33
MONTANA.					
Charles Pryse *b* ...	McClellan Gulch	1 mile	5	5	0.61
OREGON.					
Blue Gravel ...	Yank district ...	200	8	10	6.25

b Drift mines.

II. Construction of the Sluice.

The sluices which connect the bank with the chimney are of variable length and size. When the mine is first opened they are made in the top dirt, but when the bedrock is reached they are made permanently in it and carry the dirt from all the working faces to the chimney. They are ground flumes which, as they carry gravel as well as water, must be very solidly built.

When the tunnel is completed and the shaft and chimney are connected the sluices are laid through the tunnel and continue from its end to the dump. Occasionally the tunnel is itself the sluice, but this is sometimes objectionable on account of the difficulty of cleaning it up. The sluices below the tunnels are constructed like all the others. Generally the sluice is of wood and is built in the most substantial way, but so arranged that the pavement can easily be handled when the clean-up is to be made. The sides of the sluice, as they are subjected to continual wear, must be lined with 2-in. plank put in in such a way that it can be easily removed without danger of injury to the sides, and when it is worn out is replaced by new lining without changing the sides. The sluice as a whole is a composite structure, and is made up of several parts, each having different functions, but all belonging to the sluice, but placed in different sections of it according to circumstances. They are known as drops, grizzlies,

and undercurrents. The object of all these structures in the sluice is to break up the cement and gravel by interposing sudden falls, and by causing the large pieces constantly to grind against each other, to separate all the large stones and unbroken material, and finally to give both the coarse and fine particles of gold opportunity to settle and be caught in the quicksilver. The principal requirements for the sluice boxes is a sufficient grade, and a construction of such a character that the structure will be permanent, but at the same time will admit of being moved if necessity requires.

It often happens that in some of the mines the engineer cannot adopt the kind and size of sluice which would give the best results, and is constrained to resort to the most practicable makeshift. Many of these sluices look from a distance like simple water troughs supported on stilts, and seem as if they would be blown away by the first high wind, as, for instance, parts of the Magenta flume, in Nevada County, California. So far as the appliances for saving the gold are concerned, they have developed a very high degree of skill in all the hydraulic mines, and with proper facilities the working is almost invariably successful, even in poor ground. A type sluice should have plenty of fall, abundant water, and abundance of tail deposit. These conditions, however, rarely occur together in one place, and the best that can be done under the circumstances, will be the plan which the engineers will be obliged to adopt.

In laying out the grade, care must be taken to provide for the undercurrents, drops, and grizzlies. If there is a plentiful supply of water at a cheap rate, this in itself will make up to some extent for a deficiency of grade; but if water is dear, and consequently limited, a high grade is absolutely indispensable. A large amount of water compensates to some extent for low grade, and high grade supplies in a limited way the deficiency of water. But it is never advisable to have the grade higher than is absolutely necessary for the transportation of the gravels through the sluice. It is always desirable to avoid extremes. They usually have a grade of from 6 in. to 12 in. in 12 ft. Some mines are worked with as low a grade as 3 in., others as high as 9 in. to each box, and many prosperous mines work with a grade of 4½ in. to 5 in. It

depends generally on the abundance and cheapness of the water supply. If the gravel contains a large number of boulders, as high a grade as 18 in. in 12 ft. is required, but this is an exceptional case; 6 in. every 12 ft. is an excellent grade, but some of the best mines work upon 4 in., 4½ in., or 5 in. A 4 per cent. grade is, however, generally used, but in case of clay the grade is increased to from 6 to 8 per cent. Very light gravel will sometimes require not more than 1½ per cent. This is, however, too low. In general terms, ordinary gravel needs 4 per cent. and coarse gravel 6 to 7 per cent.; the heavier the gravel, the greater the grade, the more water will be required. A 5-in. grade with a number of undercurrents is preferable to a 6-in. grade without them. When the position of the sluice beyond the mouth of the tunnel has been determined, and the necessary grade for it been provided, sills at least 4 in. by 6 in. are laid 4 ft. apart; for a single sluice these are about 8 ft. long, for a double sluice they should be 15 ft. long. Posts 4 in. to 6 in. square and 3 ft. to 3½ ft. long are then placed in position, and braced with pieces of plank 1½ in. thick. A piece of board 8 in. wide is then nailed on the sill from post to post to serve as a footway. This not only allows of easily constructing the rest of the sluice, but also materially strengthens the structure. The sides and bottom of the sluice are now placed in position, and are made of 1½ in. plank and 12 ft. long. Where the planks come together they are usually grooved and filled with a tongue to prevent the loss both of water and mercury. Sometimes they are only made tight by battens nailed over the cracks on the outside.

The size of the sluices is determined by the quantity of water they are to use. They generally vary from 4 ft. to 6 ft. in width. A sluice 4 ft. wide and 2½ ft. deep, with a 2½ per cent. grade, will deliver 1200 to 1500, and the same with 4 per cent. grade will deliver 2000 miners' inches, one 6 ft. wide and 3 ft. deep, with 4 to 5 per cent. grade, will deliver about 3500 miners' inches.

The undercurrents will in some cases make up for a deficiency of grade, so that the lower the grade the larger the number of undercurrents must be. The tunnel sluice itself will constitute the first set of the sluice boxes, which may be either single

or double, according to the size of the tunnel, and should be arranged in such way as to carry the largest possible quantity of gravel. When very large quantities of gravel must be worked the sluice is made double, resting on the same sleepers, with gates between them, so arranged that the whole supply can be cut off from one and thrown into the other indifferently, or that both may be made to work at the same time. If any part of the sluice should need repair during the run, these gates are used, the water being simply turned off from a section between two gates. Sometimes when it is known that repairs are likely to be necessary, a side sluice is built, or the sluice made double, and provided with gates in that section. When the sluice is running full of water, these gates are held so tight by the pressure that they are very difficult to open. A windlass and ropes are always provided so as to move them easily. Such an arrangement of gates allows one sluice to be always clearing up while the other is being worked.

The water running in the sluice should be deep enough to cover the largest boulder which would be likely to be sent down, which makes a very great difference in the quantity required to wash coarse and light gravel. As far as the yield of the gold is concerned, it necessarily follows that low grades and a small quantity of water yield the most gold. The amount, however, cannot always be controlled, as whatever comes must be washed. The danger of using a large quantity of water is packing the riffles.

The length of these sluices will be very variable, and will depend on the nature of the gravel, and no general rule for determining it can be given. It must be decided in each case by trial, though the approximate length is easily determined beforehand. It must be sufficient to break up the gravel and to allow of the fine gold being caught by the mercury. If, during the working, assays should show too much gold in the tailings, the sluice must be lengthened. As the velocity of the current is high, if the sluice was perfectly straight, the gravel would be likely to be carried along with but little change. It is for this reason never made perfectly straight, but is curved in a general direction bending from one side to the other. Whenever a curve

is necessary it should be made of the largest practicable radius, and the sluice should be raised upon the outside, from ½ in. to 1 in. according to circumstances. This checks the force of the current on the outside, and distributes the flowing matter evenly over the bottom of the flume, and prevents the wearing of a deeper channel in the pavement on the outside of the curve.

When the bottom and sides of the sluice have been constructed it is ready to receive the pavement and lining. The pavement is made either of the stones of the country, of blocks of wood, or of iron. When stones are used, oval ones are always selected. The rock generally used is basalt, because that is the most easily accessible, but quartz or any other rock may be used, care being taken to have the stones as nearly uniform in size and shape as possible. The stones are placed edgewise, Fig. 29, with a slight

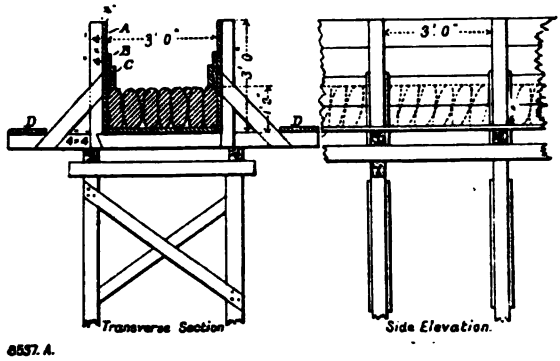


FIG. 29. DETAILS OF SLUICE WITH STONE PAVEMENT.

slant in the direction of the current. They are generally from 9 in. to 12 in. thick, as there is considerable interval between the stones; this is first filled up with gravel in the preliminary working. The interstices between the top stones serve as depressions in which to catch the amalgam. This pavement is placed in compartments 6 ft. to 8 ft. long. These are held in place by strong pieces of plank which are securely fastened across the sluice at the distances which regulate the length of each. This secures the sluice against the accident of having any part of the pavement get loose, which might occasion serious damage to the sluice, and makes it possible for any one compartment to be taken up and repaired at will independently of any other. As this pavement

for a double sluice weighs about 8 tons for every 10 ft., the necessity of constructing the woodwork in the strongest way will be seen. Above the pavement a side lining B of 2-in. plank, or worn-out wooden pavement blocks, to the height of 10 in. to 15 in., is nailed, to protect the sides against the wear of the gravel. This side lining will frequently project 2-in. below the rock pavement. Upon this a false side lining C, from 6 in. to 8 in. wide, is nailed, which comes down upon the outside layer of stones. Such a pavement is very durable. It however requires much more water and a heavier grade than a wooden one. It is also more economical than any other kind, but the labour of handling the stones at the time of a clean-up is very much greater. They are generally placed in parts of the sluice that do not require to be cleaned up very often. In large mines, like the Spring Valley, they last almost twice as long as wooden blocks, and in some of the other smaller mines they last for an indefinite period.

The tunnel sluices, and even the main sluices, are paved with

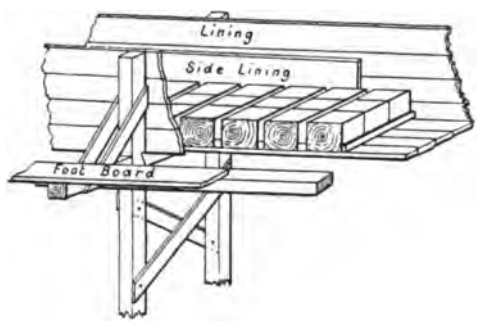


FIG. 30. SLUICE PAVED WITH WOODEN BLOCKS.

wooden blocks. This pavement is made of blocks of heart sugar pine, and is shown in Fig. 30. The blocks are usually 12 in. to 18 in. square, cut across the grain, and 6 in. to 12 in. in height. They are sometimes used with the corners cut, or, more rarely, the natural section of the wood is used. They are laid across the sluice, and secured by a plank 1½ in. in thickness, and 3 in. to 6 in. high, which is nailed to the lower part of the block. These nails are without heads, and project about 1 in. on the face side. Another tier of blocks is now placed in, and driven on to the projecting nails, so that they come close to the cross-piece.

Another cross-piece is then put in, and in this way the bottom of the sluice is filled. Occasionally they are held in place by soft pine wedges, in which case the blocks must be square. The side lining is now brought down to the tops of the blocks, and holds them in position; there will then be, between each row of blocks, a space corresponding to the cross-piece 1½ in. in width.

In very large mines like the Spring Valley Mine, such a pavement will not last over three months, but often in small mines they last through a whole season, and where they are not too much worn they can even be turned over and thus made to last during two seasons. When they are worn down to 4 in. or 5 in. they are no longer serviceable. Sometimes, when the sluices do not require to be cleaned often, alternate sections of block and rock pavement are used. The wear of the blocks is in this way considerably reduced. Such pavements are not so easily replaced as the wooden ones, but more easily than rock. They have given good results both in yield and wear where they have been used. When there is a large amount of heavy material, as stone, washed from these sluices, it is necessary to have the pavement made of riffles, which run not across but lengthwise of the sluice. They are often made of wood, and when the quantity of boulders and pebbles to be transported is very great they are protected by straps of bar iron fastened to the top. Such a sluice 3 ft. 8 in. wide, with a grade of 18 in. in 12 ft., was used at the Paragon Mine in California. In some cases where there is a limited or very little fall, a great advantage will be gained in using riffles made only of iron, as the smooth surface reduces the resistance to a minimum, while the spaces for catching the valuable metals can be largely increased. Old railroad rails have been successfully used as riffle bars; they are usually the best when they can be had. These are laid close together and are easily placed and easily retained in position, for which reason they are generally put in at the head of the sluice, where clean-ups are most frequently necessary. If strap iron has to be used, it is not quite so advantageous as rails, though necessary where large quantities of small boulders are constantly passing through the sluice, and rails cannot be had.

The advantage of this construction is shown by some experi-

ments made at the Morning Star claim in California, where three sections of flume, each 65 ft. long, were laid at a distance of 300 ft. from where the bank was being worked. The first section was as usual laid with blocks, the second with old rails, and the third with blocks. When the clean-up was made the middle section gave 9 oz. more gold than both the others combined.

Of these three varieties of pavement, the wooden is expensive in construction, presents the least surface for catching the amalgam, and lasts but a short time, generally not more than three or four months. On account of the rapid wear, the gold caught in the mercury is likely to be ground out of the pavement and carried away. The stone is the cheapest, so far as the material is concerned, but is expensive to take out and replace. It has a very irregular surface, and, to a given area, has the maximum number of interstices in which the amalgam may lodge. It does not wear as rapidly as the wood, and is a very advantageous construction outside of the tunnel, where there is plenty of space to pile the stones during a clean-up. It is even a question whether, all things being considered, it is not better than wood.

In most of the mines it is usual to have all these different kinds of pavements in different parts of the sluice, the particular form to be used depending upon the kind and quantity of the gravel and the fineness of the gold. In some few cases the tunnel is not paved at all. In the North Bloomfield Mine the irregular slates at a high angle through which the tunnel was run were used for and answered all the purposes of riffles for the last part of the sluice, but this is an exceptional case. When about eight million cubic feet of gravel had been washed through this tunnel the slate was found to have been worn down about 16 in. After between 22 and 23 million cubic yards had been treated it was found to have been lowered about 3 ft., and in places to have been worn into deep holes, which, however, did not seriously interfere with the use of it.

The expense of constructing a double sluice, independent of the pavement, depends, of course, upon the price of timber. Where it can be had for $20 a thousand, and no blasting is required in constructing the bed, it can be done for $2 per foot.

A single sluice 6 ft. by 3 ft. can be constructed for $1.25 per foot.

A. The Undercurrent.

Undercurrents C, Fig. 31, are large flat areas of irregular shape, varying from 10 ft. to 35 ft. wide, and 30 ft. to 100 ft. long, of a considerably less grade than the sluices. They are intended to separate the stream in the sluice, so that the under part, which contains coarse material, may pass on in the main sluice or be drawn over the side, while that portion of it which contains the gold and the fine sand is separated by passing through a grating into a very broad sluice of much lower grade, receiving a supply of clean water, where, on account of the diminished velocity of the current, the gold and float amalgam can settle. The tails of the undercurrent are returned to the main sluice. These undercurrents are placed in the sluice at variable distances from the outlet of the tunnel, according to circumstances, and are usually from 75 ft. to 100 ft. apart. The shape of the undercurrents will depend entirely upon the shape of the ground where they are to be placed. They are generally parallelograms, but are often triangles, or of irregular form. They are usually 20 ft. to 40 ft. long and 40 ft. to 50 ft. wide and about 15 in. deep. The chief thing is to have a space of 600 to 2500 square feet, the object of which is to catch as large a quantity as possible of the fine sand which is being carried down, and to treat it outside of, and independent of, the sluice. This is done by interposing in the bottom of the main sluice, apertures 20 in. to 40 in. in width. A cast-iron frame is placed in this opening, into which fifteen to twenty steel bars 1 in. by 4 in. and 1 in. apart set on edge are inserted. Beneath each of these iron gratings an under sluice is placed at right angles to the main sluice, and leads to the large platform called the undercurrent. The small gravel and sand containing gold, which flow along the bottom of the main sluice, drop through these bars, are caught in the sluice box below, which has an incline of about 1 in. to the foot, and are carried with considerable velocity over the head of this large platform, which, as it is very wide as compared to the sluice, is provided at its head with check

boards to distribute the water and gravel as evenly as possible over the whole surface of the undercurrent, and also with all varieties of riffles. These are arranged in every possible direction, so as to interpose obstacles to the velocity of the current, and make the sands rest on the mercury and amalgam. They are always arranged in such a way that, by the removal of a few slats, all or any part of the riffle bars can be easily taken up. The undercurrent itself has a grade of about 10 in. to 12 in. in 12 ft., and empties into the main sluice at its lower end. As the undercurrent draws a considerable quantity of water from the main sluice A, Fig. 31, this is narrowed generally from 6 in. to 1 ft. the whole length of the undercurrent, and after passing it are continued the same width as before.

These undercurrents are sometimes placed alternately on the sides, sometimes on the same side, and sometimes on both sides of the sluice, according as the exigencies of the case, or the lay of the ground demand. When placed on both sides, the sluice will have to be narrowed twice as much. This is specially necessary in those cases where there is not sufficient grade. The diminution in the velocity of the stream makes it possible to catch on the undercurrent a considerable quantity of the rusty and fine gold which would otherwise be carried along in the main sluice and perhaps lost, or at least not so quickly caught. At the American Mine, in Nevada County, there are twenty undercurrents. In a few exceptional cases there are as many as twenty-five. Gold is caught in the last one, and there is but little doubt that, if several more were placed at the end of the last one, gold would still be caught. It is well known that diamonds exist in some of the gravels of the deep placers. They are very seldom found, and are not likely to be found except in the last of the undercurrents, where the force of the water is not too great to carry them off.*

For undercurrents iron riffle bars 1¼ in. wide, 1 in. thick, and from 12 ft. to 14 ft. long, are fitted into cast-iron cross-supports, which serve to hold them in place and allow spaces of ¾ in. between each. These cross-supports are slotted so as to hold the longitudinal bars firmly, and are made about 2½ ft. long.

* Prod. Gold and Silver in the U.S., 1882, p. 732.

This form of riffles will be found advantageous in cleaning up, as they can be easily handled, and their wear can be made even by turning them over, and end for end alternately, in each succeeding run. The cost of such riffles for an undercurrent 10 ft. wide, 48 ft. long, will be less than $500 in San Francisco, and by their use a large saving of gold can be effected.

It has been suggested that all the material contained in the gravel which is washed through the tunnel should be sized by siftings through steel bars, until, after a certain length of sluice was traversed, these bars should not exceed two or three diameters of the coarse particles of gold found by experience to occur in the mine. Such light material could be carried on with a very slight grade with a small volume of water, thus making the sluice practically a continuous undercurrent below the point where the last of the coarse material had been thrown out. Thus by a series of very flat undercurrents the tails could be returned to the main sluice. Additional streams of clear water could be used to offset the loss of it over the grizzlies. The size of pavement and the length of the sluice would then be settled by continuous experiments and experience in each mine. This suggestion seems to be an excellent one, but has never been carried out on any very large scale. There is no question but that the rapidity of the stream of water, and the force necessary to move the heavy barren materials through the sluice, carries off with it not only very much fine gold, but also the gold which the miners call rusty, breaks up the amalgam or causes it and a considerable amount of floured mercury to be lost. A large number of undercurrents corrects this defect to a considerable degree, but the very fact that it has been found advisable to use as high as twenty-five undercurrents shows how much could be gained if such a system as this could be carried out. Whatever the number of undercurrents have been in a mine, gold has always been found below the last one. Unfortunately, the topographical condition of the country is such as will seldom allow of such a type sluice being erected, besides which, the large amount of capital required to put up such a sluice as this generally prevents the engineer, who is always forced by limit of time and

the desire of stockholders to get immediate gains, to put up something less costly that will bring in quick returns, from attempting it.

B. The Drop-Box.

Generally the undercurrent and main sluice discharge into what is called a drop-box. This drop D, Fig. 31, is several times wider than the sluice, and is built of very heavy timber, framed so as to resist great pressure. The height of the fall will, in many cases, be regulated somewhat by the length of the undercurrent, which generally discharges into it directly over the sides of the drop; but in others, it is determined by the lay of the ground, so that the undercurrent will discharge from an end sluice. Where a fall of 40 ft. to 50 ft. can be found, the drop is placed directly under the main sluice, which is continued from one side either at right angles to its former direction, or, if the lay of the ground will permit, in the same direction, the drop-box being only the continuation of the sluice at a lower level.

As the drop-box is destined to be a permanent structure, it has a very heavy stone pavement which is 4 ft. to 5 ft. or more below the level of the main sluice. The bottom of the drop is considerably deeper than the continuation of the main sluice, in order to catch any heavy material such as gold and amalgam.

The object of the drop is as far as possible to break up pipe-clay and disintegrate cement. Even if the gravel is apparently quite loose and running free, a number of such drops can be usefully introduced if there is sufficient fall, and with tough clay or gravel they are extremely valuable. They should be not less than 2 ft., and as high as the available grade will admit.

C. The Grizzly.

The continuation of the main sluice B over the drop-box D, Fig. 31, is called a "grizzly," and is only applicable when the sluice is near a precipice of considerable height. Its object is to remove from the sluices the large sterile matter which the current carries along with it, but which only serves to encumber the sluice, wear out the pavement, and break up the amalgam. To provide for it, the sluice is continued as usual until it projects a

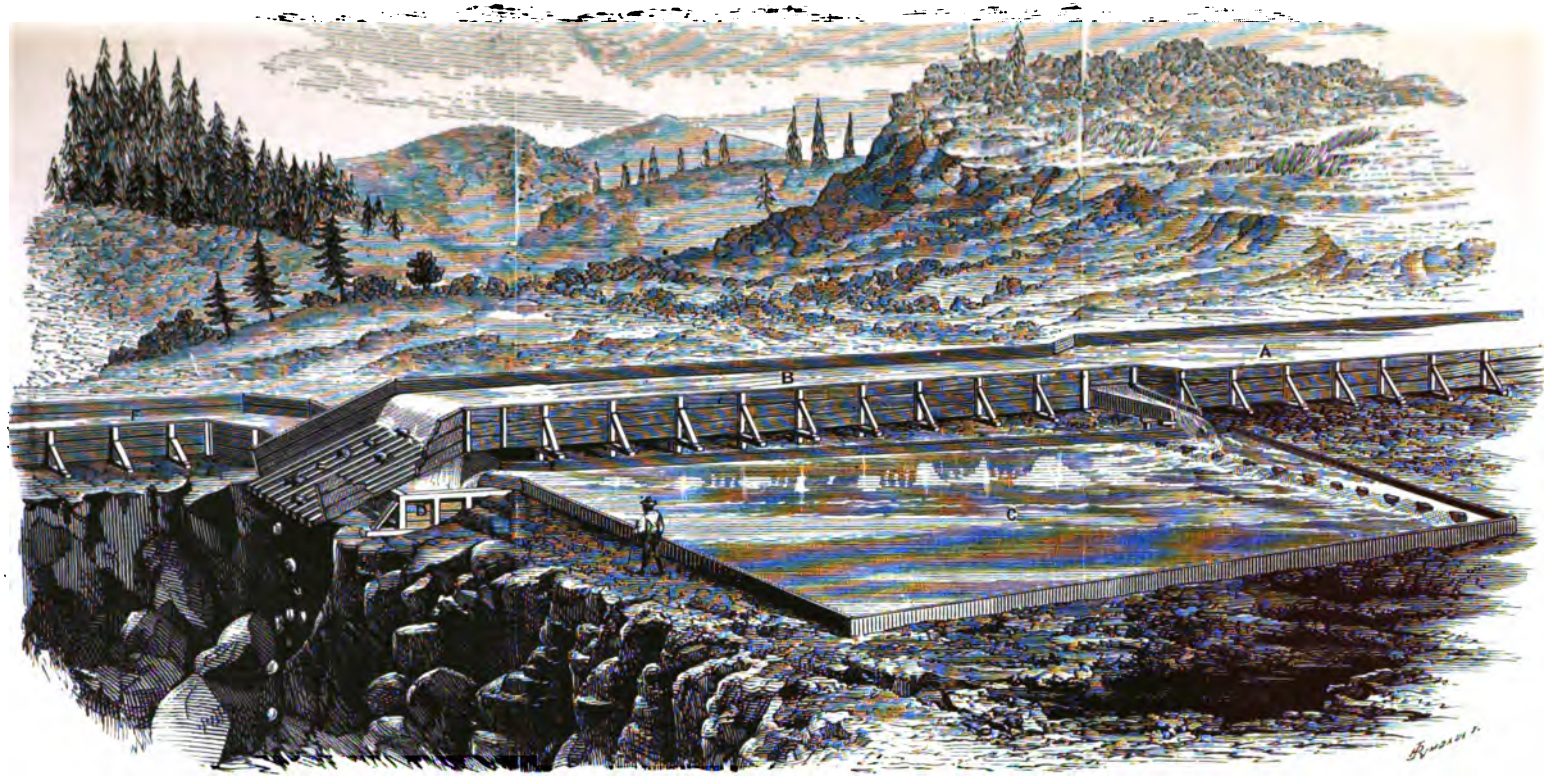


FIG. 31. A, MAIN SLUICE; B, CONTRACTED SLUICE; C, THE UNDERCURRENT; D, THE DROP-BOX; E, THE GRIZZLY; F, CONTINUATION OF THE MAIN SLUICE.

little over the drop-box. An inclination of 25 deg. to 30 deg. is then given to the sides, which are carried completely over the drop-box, projecting far enough beyond the precipice to insure the discharge of the stones over it. The bottom of the sluice from the point where the sides begin to incline, is made of railroad rails placed 6 in. apart. All the material carried by the sluice comes over the drop-box, and all the water and the gravel of less size than the distance between the rails, falls through the bars into it; while the stones are caught and easily slide over the precipice. By interposing at given distances a number of grizzlies and properly regulating the distances between the bars, the large boulders can be separated at the top of the sluice and the small ones next and so on, until only the gravel containing the gold is carried forward by the water. By this simple contrivance a large amount of useless material is got rid of.

In some of the sluices swinging plates are used to catch the float gold, which are the width of the sluice. They are curved sheets of copper, silver plated and amalgamated, and dip into the water several inches. They are swung in the sluice through eyes and fastened to their sides with wires.

When the claim is not being worked, it will be most economical not to let the sluice stand dry. A small stream of water should be kept constantly running through it, so as to keep the joints tight. When allowed to dry the cracks open, and may be a serious cause of loss, both of mercury and amalgam, when the claim is opened again for working.

These undercurrents, drops, and grizzlies are repeated in the course of the sluice as often as the lay of the ground will allow, or the judgment of the engineer may dictate. Beyond them the tail sluices are continued to some stream or ravine, where the refuse may be deposited. This part of the sluice demands good judgment in its location. It must be put up so that it can be easily added to, taken down, or its direction changed if necessary, and yet the location must be made with a view of being permanent. The enormous amount of material to be deposited demands the greatest foresight, and also an accurate knowledge of engineering principles, since it is not always possible to tell beforehand exactly what effect the sand will have, and

exactly how it will be deposited. This is more particularly true when several claims "tail" the one over the other, as they sometimes do. When such dumps give out, it is always the lowest claim that is closed first; if owned by the same company the upper ones do not work while the lower claim has a dump. It may be possible, by constructing a tunnel, to carry the tailings into ravines from which the spring freshets will clear them, and this should be done if no other outlet for them can be found. The usual way of disposing of the tailings is to deposit them in some large river, which will generally carry them off.

To give an idea of the enormous mass of material required to be moved, it has been stated* that at the Sailor's Union Mine, Iowa Hill Ridge, 1,615,577 parts of detrital material had to be washed away to obtain one of gold, and at the North Bloomfield Mine it required 12,107,116 parts to be washed for one of gold obtained. The accumulations have gone on to such an extent that Bear River, which formerly had a depth of 80 ft. in its channel, is now so full of detritus that the tall pine trees which formerly grew on its bank show only their top branches just above the surface of the accumulated tails. In order to avoid apprehended difficulties, some of the companies adopted the policy of purchasing the ranches on the line of the outlets of their tails. One of them purchased as much as 12,000 acres, but it was of no avail to prevent the disastrous litigation which has culminated in closing so many of the claims. The Tuolumne River,† which was formerly a stream over 500 ft. in width, with a channel 14 ft. deep in the centre, and a fall of 18 ft. to the mile, was in twenty-one months of 1873 and 1874 entirely filled up, so that the water hardly ran over the bed of the river, and mining had to be abandoned. In 1875-6 the freshets washed the river clean, since then 261,000 cubic yards have been "hydraulicked" into it, and the stream in 1876 was only 31 ft. wide and 1 ft. deep. The Spring Valley Mine, in order to impound its tails, spent $461,435.87 for land and $270,811.48 for its débris canal. Much of the land has since been sold and about half its

* Prod. Gold and Silver in the U.S., 1882, p. 730.

† Trans. Am. Inst. Min. Eng., vol. vi., p. 39.

DIMENSIONS, OF SLUICES IN PLACER MINES.*

Name.	Location.	In Tunnels or Cuts.				Outside of Tunnels.			
		Width.	Grade.	Length.	Style of Riffles.	Width.	Grade.	Length.	Style of Riffles.
		ft.		ft.		ft.		ft.	
Black Hawk.	Five miles north-west of Quincy, Plumas County.	...	... ...	...	... ...	1⅔	5 in. in 12 ft.	½ mile	Blocks.
Del Norte.	On Klamath River, near Happy Camp.	2	8 in. in 12 ft.	300	Blocks.	2⅔	,, ,,	1200	,,
Gold Run.	One mile south-west of Gold Run, Placer County.	6	6 in. in 12 ft.	2500	Old car rails.				
Hager and Hass.	In Junction City, Trinity County.	...	... ...	...	... ...	4	3½ in. to 7 in. in 12 ft.	1440	,,
Harvey Bros.	Weaver district, Trinity County.	...	... ...	...	... ...	3	3 in. in 12 ft.	4800	,,
Holman and Mahon.	Do. do.	...	... ...	...	... ...	3⅓	2 in. in 12 ft.	3600	,,
John Young.	In town of Oro Fino, Siskiyou County.	4 to 5	⅞ in. to 1½ in. in 12 ft.	2000	Blocks.				
Oroville.	On Feather River, near town of Oroville.	...	... ...	...	... ...	4	2½ in. in 12 ft.	2400	,,
Rule Bros.	Weaver district, Trinity County.	...	... ...	...	... ...	3	,, ,,	3000	,,
Spring Valley	Cherokee Flat, Butte County.	6	8 in. in 16 ft.	1800	Basalt.	4 to 6	4 in. to 8 in. in 16 ft.	6½ miles	Basalt and wdn. blocks.
Van Emmons	Town of Michigan Bluffs, Placer County.	2⅔	13 in. in 12 ft.	3600	Wooden blocks.	2⅔	10 in. in 12 ft.	300	Rock.
Weed's Point.	Two miles north of Camptonville, Yuba County.	4½	6½ in. in 12 ft.	600	Rock and wdn. blocks.	3⅓ to 4	4 in. in 12 ft.	4200	Blocks.
Wiltshire.	Trinity County.	...	... ...	...	... ...	4	6 in. in 12 ft.	1200	,,

* Report of U.S. Census for 1880, vol. xiii., p. 197.

value recovered. The débris flows through a canal thirty-two miles long to two retaining dams 1800 ft. wide and over 12,000 ft. of tule land.

There would be great danger of the rivers being entirely obstructed from the deposits of such enormous amounts of detrital matter in their beds, but such danger is usually remedied by the annual freshets, which keep the river channels moderately free. Experience has shown that about every decade there comes a freshet such as sweeps the channels of the rivers about clean. But, nevertheless, the continual deposits over a period of thirty years on its bed have so raised the bottom of the Sacramento River that a system of levees are in some places required to protect the surrounding country from being flooded during the rainy season.

The Table opposite gives the principal dimensions of some of the prominent sluices, and the character of their pavement.

The shafts, tunnels, sluices, and dump having been provided for, the mine is ready for working. This may be done by water alone when worked on a small scale; but when on a large scale by water and powder together. In both cases the supply of water should come from the greatest possible elevation.

III. Bank Blasting and Cutting Down the Banks.

Bank blasting on a large scale is an operation which requires the greatest judgment and experience. It was invented by J. F. Pierce, who introduced it in the year 1860 at the Blue Point Gravel Mine, near Smartsville, Yuba County, California. Its object is to break up the gravel and cement so that it may be more easily piped. It was used in a very small way at first, but with such satisfactory results that it is now everywhere used, and on a very large scale. Previous to its introduction the banks were undermined with picks, and not unfrequently they fell without warning and buried the men, or gave them so little time to escape that they were caught and seriously injured. The height of the bank, the tenacity of the ground and its general structure vary so greatly in different places, that the amount of explosive material and the best methods of arranging it are matters for discussion in every particular

case, and are not subject to any general rule. It is a matter that is worthy of more careful investigation than has yet been made of it, especially as to under what circumstances it is best to use or abstain from using high explosives. When the ground is very hard or compact, so that the jet does not easily affect it, or when the banks are worked so high that the nozzle cannot with safety be brought near enough to the bank to do good work, for fear of sudden caves and dangerous runs of the water and gravel, or when it is desirable to wash a large quantity in a very short time, the ground is blasted for the purpose of shattering the bank, so that the loosened earth may be washed away with a less expenditure of water and time. The object of the powder is simply to make the work of the water more effective.

Bank blasting, as it is called, may be done from shafts or from drifts. When the ground to be broken up is underlaid with cement, or when the depth of the gravel is so small that drifting would be too expensive, shafts are sunk, the bottoms of which are excavated in the form of a bottle, into which five or six kegs of powder are placed, the shaft thoroughly tamped and then exploded. If the shaft is 15 ft. to 20 ft. deep, this will shatter the ground for an area of 15 ft. or 20 ft., so that it can be easily piped. When done from drifts it may be of almost any magnitude, the length of the drifts depending upon the quality of the ground to be broken. When done from shafts, they are necessarily more or less limited. In the former case one or more cross drifts, depending on the size of the blast, are always run across the end and middle of the main drift; the end one should be about half the length of the main drift; the auxiliary, if there are only two, may be placed about half way, and will be about one-third its length. Occasionally the cross drifts are turned on themselves, so as to have the shape of the letter T. The sum of the lengths of all the cross drifts should be about equal the length of the main drift. The drifts themselves are made just as small as the men can work in, and are generally 3 ft. by 4 ft. The length of the main drift will usually be for low banks from one to one and one-half times the height of the bank to be blasted. For very high banks, about three-quarters of the height is sufficient.

All sorts of explosives, such as dynamite, giant powder, &c., have been successfully used for bank blasting. Up to within ten or twelve years black powder was generally the favourite, on account of the ease and safety of transportation. A slow burning powder is generally preferred, as the object is simply to lift and loosen the ground. As much as 2500 lb. of giant powder have, however, been used in a single blast, and from 75,000 to 100,000 cubic yards of earth broken up so as to easily yield to the nozzles. When it is used, however, each separate lot must be carefully tamped. Though much more expensive than powder, it is calculated that 1 lb. of it will produce as much effect as 5 lb. to 6 lb. of black powder. High explosives are generally used for blasting the boulders of rock or of pipe-clay and tree stumps brought down by the undermining of the bank, either when they are in the way or when they cannot be conveniently removed by cranes. The quantity of powder to be used depends on a variety of circumstances. These are the height of the bank, the resistance of the gravel, and the quantity to be blasted at a time. It is generally calculated that it will require three-quarters of a keg of powder for every thousand cubic feet of gravel to be broken in the bank. If enough is used to blow out the bottom, which is sometimes the case when the bank is at least 100 ft. high, the whole mass, falling the length of the drift, will be disintegrated and crushed. When it is not blown out the whole mass of gravel will be raised a little, and will be filled with cracks in settling back, so as to be easily piped.

The amount of powder used in single blasts varies from 20 to 1000 kegs, the exact quantity depending on the amount of gravel to be moved. It is always better to allow a larger quantity than a smaller one, because if the blast should fail the expense of an excess of powder would be less than that which would be caused by the loss of the whole of it. The usual method of handling the powder is to run a small drift into the bank at or near the bedrock, and from it two or more cross drifts parallel to the face of the bank in the form of a T. Powder is placed in the cross drifts in kegs or boxes in rows, and after connecting the open kegs with wires, which are buried in the powder, a bulkhead of timber is placed across

the main drift, and it and the side drifts are tamped. Sometimes when it is necessary to use very large quantities of powder a more complicated system of cross drifts is adopted, and accessory short galleries, called "lifters," are used, which are driven from the extremities and centre of the cross drifts towards the face of the bank. These modifications are used to increase the efficiency of the blast. When the bank is 80 ft. to 120 ft. high, the main drift should be about the length that the bank is high. It should be as small as it can well be worked, about 4 ft. high and 3 ft. wide. For such a drift as this about 600 kegs of powder, containing 25 lb. each, will be used, two-thirds of which will be placed in the cross drifts at the end of the main drift, and one-third in the auxiliary drifts, placed about half way. If the main drift was only 80 ft. to 85 ft. long, 400 to 500 kegs of powder would be used; if the banks were 250 ft. high, from 1500 to 2000 kegs would be required. The tamping of the drifts requires the most careful attention. It must be done in such a way that it will resist the force of the gases produced, otherwise it might blow out, and besides burning the powder to waste, endanger life. The material taken from the drifts is used for this purpose. It should be free from large stones, which would be dangerous in case it should blow out and moist enough to pack. It should be tamped against the sides and top to insure that there will be no outlet for the gas at these points. When fuzes are used it is generally best to tamp most of the main drift. When the blast is fired by a battery less tamping is done. When electricity is used double and triple wires are provided, and to insure against failures the connections must be very carefully made.

When working from the side towards the centre of the deposit the banks often become so high that it is neither practicable nor safe to pipe down the entire bank in a single operation. In such case the height is divided and the claim piped in benches, the upper part being entirely run off before the lower is attacked. This will be generally the case when the bank gets to be over 150 ft. in height.

When the mine is worked in benches, very powerful blasts are not necessary. Shafts are sunk to the depth of 10 ft. to

20 ft., which are either simply enlarged at the lower part so as to be more or less bottle-shaped, or short drifts are run from it at the bottom. These are generally used for small work. The charge of powder is correspondingly light. In blasting very high banks it is the general rule within the last few years not to attempt to raise and shatter the whole mass from the bottom upwards by the direct action of the explosive, but rather by means of cross drifts driven parallel to the face of the bank at such distances that the strip of ground between them and the face can be blown out. The power which is thus used, instead of being sufficient to raise the whole bank, as was formerly the case, simply blows out the lower part and allows the bank to fall by its own weight.

Very large blasts have been fired in some of the California mines. At the Spring Valley Mine, Butte County, a blast of 1000 kegs of black powder was used. In May, 1880, 25,000 lb. of Judson powder were effectively exploded in this same mine. At the Enterprise Mine a blast of 1700 kegs was fired in a bank 250 ft. high. Bank blasting not only facilitates the work of the nozzles and renders it possible to work the tough ground which would otherwise be moved with difficulty by the water or even perhaps be entirely unworkable, by breaking up the cement, pipe-clay, &c., but it serves also the very important purpose of disintegrating the gravel and freeing the gold particles. The more thoroughly the ground is shattered the better it is for sluicing. In the year 1875* a blast of 17,500 lb. of powder was exploded at the Paragon Claim, Placer County, to break up a bank 150 ft. high. The drifts were 4½ ft. high by 5 ft. wide; their total length was 325 ft. They were arranged in the shape of a T. At the end, to the right of the main drift, which was 110 ft. long, was a drift 70 ft. long, and at its end another one parallel to the main drift 55 ft. long. To the left the drift was 60 ft., and at its end one of 30 ft. long parallel to the main drift. In the drift to the right 400 iron kegs of powder were placed, and in the opposite side 300 kegs. The mouth of the drift was tamped 75 ft. from the end, and the T drifts were tamped 10 ft. each way, which, with the width of

* Min. Com. Rept., 1875, p. 107.

the drift, made 100 ft. of tamping. A large amount of space was left in the L drifts for the expansion of the gases, so as to render the action of the powder more effectual. The bank was 140 ft. high. The electrical machine with which the powder was fired was 450 ft. from the mouth of the tunnel. The total length of wire was 1500 ft. The running of the drift cost $300; the powder, &c., $2700. The amount cleaned up after the blast was $200,000. A blast of 2000 kegs, or 50,000 lb., of powder was exploded at the Blue Point Gravel Mine, December 29th, 1870. The main drift for the blast was 275 ft. long.* To the left of the mouth there were six galleries commencing 75 ft. from the entrance, each 120 ft. long; to the right six others, each 80 ft. long. On the first drift to the right was a drift parallel to the main drift 15 ft. long. At the end of the first drift also there was another 15 ft. long parallel as before. Each one of these drifts was 3 ft. by 4 ft. They were called lifters. The powder was equally distributed through the drifts. The blast was fired by electricity. There were ten points at which the powder was exploded at the same time. This blast raised 150,000 cubic yards of earth vertically from 6 ft. to 10 ft. In 1877, 11,000 kegs were fired at this same mine. In 1872, 3500 lb. of giant powder were fired at the Harriman and Taylor Claim at Gold Run, Placer County, which disintegrated 200,000 cubic yards of gravel. In January, 1879, the Nevada Reservoir and Ditch Company exploded at Sucker Flat, Nevada County, 1000 boxes, or 50,000 lb., of Judson powder, equal to twice that quantity of ordinary powder, loosening up completely a gravel bank 200 ft. high, out of which $150,000 were washed. In November, 1875, 36,000 lb. of Judson powder was exploded at the Dardanelles Hydraulic and Drift Mine, Placer County, which broke up 500,000 cubic yards of cement. The outside face of the bank was 1000 ft. long and 175 ft. high. Into the face of this bank five parallel drifts were run, across each of which two or more galleries were cut at right angles. The total lengths of all the drifts was 1200 ft. The powder was distributed in lots of from 1000 lb. to 2500 lb. in twenty-eight different chambers, in each of which were three exploders. These were fired by an electrical

* Min. Com. Rept., 1871, p. 72.

machine from the outside. At the Blue Tent Mine, in Nevada County, there were exploded during the year 1880, 60,000 lb. of black powder and 4000 lb. of giant powder. Two blasts were fired during the year, consuming 25,000 lb. of black powder. In the year 1881 this company blew up a bank 200 ft. high, using for the purpose 1720 kegs of black powder, or 43,000 lb. Many other examples might be cited, but these are sufficient to show on how large a scale bank blasting has been used. All of these blasts were successful. Indeed, they rarely fail, and are relied upon as the most economical plan of disintegrating gravel previously to piping it.

The powder is generally emptied into long boxes, and is fired by electricity. When fuzes are used they are placed at equal intervals, so that one fuze corresponds to about 50 kegs of powder. In many cases, however, this is not done; the heads are simply taken out of the kegs, and a fuze inserted in each one. This method, as it multiplies fuzes, rather increases the element of uncertainty as to the combustion of the whole of the powder employed. The object of having large receptacles is not only to secure complete combustion, but also a simultaneous ignition of the powder. Where danger from water is apprehended, the boxes in which the powder is placed should be painted with tar, in order to make them perfectly secure, and the holes for the fuzes made in the top of the box, and then closed and coated with wax. Wires connecting with the battery are then placed in position, and the main drift securely closed at the first cross drift with heavy timbers, and filled up with tamped sand to its mouth. The discharge is in such a case made by an electrical apparatus.

A great area of ground can be broken in this way, when the drifts and cross drifts are arranged for the purpose. The quantity of powder should always be calculated in such a way as to do the work thoroughly. It is better to use more powder than less, as the extra powder will more than pay for itself by the loosening of the ground. Economy in this direction is generally very expensive, since too little powder would not break up enough ground to repay the cost and time of driving the drifts and cross drifts. Shafts with one or more drifts at the bottom, or sometimes simply enlarged, are

used to loosen small patches of ground, or, exceptionally, to loosen the upper gravel, when the lower stratum consists of a heavy cement. In such a case, the top if it is light enough is piped off, or if it is heavy is worked off in this way first, and the bottom then blasted with drifts. These shafts are usually 4 ft. to 5 ft. in diameter, and when the connection is made, the shaft is closed with the material which has been taken from it and thrown out at the sides.

Almost all hydraulic mines, even those which do not use bank blasting, consume considerable quantities of high explosives, which are conveniently used in damp and wet ground, and require very little care in tamping. The total amount of powder thus consumed is considerable, but for individual mines the expense is not a very heavy one. Where bank blasting on a large scale is done, the cost of explosives is a very large item of expense, and must always be taken into consideration, in connection with labour and loss of time in preparing each blast, in counting up the cost of the gold produced. In breaking up very large boulders, holes are sometimes drilled, and from one-half to two sticks of nitro-glycerine are put in and fired as usual. A stump is destroyed by boring a hole and exploding in it a part of a cartridge. Small quantities of high explosives are used sometimes for breaking up patches of cement. At Dutch Flat a system of top blasting with giant powder is used to break up the boulders. The cartridges are broken, the powder placed on the rock, covered over with clay and fired with a fuze and cap. Large boulders that would otherwise have to be drilled to be blasted are broken up in this way. Very large masses of pipe-clay are sometimes blasted with giant powder. This is done with a clay auger, which bores the proper sized hole to the centre of the mass. A quarter or sometimes a whole cartridge is inserted, which is fired by a fuze, and reduces the clay to a condition fit to enter the sluice.

Cutting Down the Banks.

When everything is ready for working, some of the upper timbers of the shaft are removed, and the water and dirt, which may or may not have to be loosened by hand, are run into it

in large quantities, but not in such a way as to choke the shaft. The ground around the shaft is shaped to a terrace or incline, so as to secure the future working of the mine without danger of choking the shaft. The ground must be shaped as rapidly as possible, so as to admit of the use of the nozzle. A great deal depends upon this preliminary opening. If it is judiciously executed, no further trouble than that which would arise from ordinary accidents need be apprehended; but if carelessly done, the future of the claim may be compromised from the start. It does not take long to work out a sufficient area around the shaft to admit of the nozzle being used with great advantage, more especially when the banks are at a little distance, and the gravel is brought down in sluices.

The opening of a mine should be made in such a way as to secure with the least possible delay a large open space perfectly free from all obstruction, and to introduce into it as many nozzles as can be advantageously used in it. To cut down a bank, if the works are large, one pipe is kept playing on the under part on one side of it at an obtuse angle, and another one at about the same angle on the opposite side. When the dirt is stiff and does not fall readily, a third is used to wash away the débris, but under ordinary circumstances two are sufficient. When the claim is small and the bank caves readily, one pipe does the cutting, and the other washes the débris away. When first thrown upon the bank the water spatters in every direction, then the stream buries itself in it, and spurts out from the sides, and little by little the hole grows larger in front, and the depth increases until the bank is undermined in the shape of an arched cavity of variable depth, depending on the compactness of the gravel in the bank. The miner knows from experience just how deep to make the cavity. The nozzle is then moved, another arch is formed, and so on, and when several have been made the walls between them are broken down and the bank caves. When the bank is about to cave the water should be turned off for fear that the run of débris should cause accidents. Whenever it is practicable caves should be made towards evening. The caved material is then washed by the night shift into the shaft connecting with the sluice boxes, care being taken to render the

flow as uniform as possible. All the large material left behind which cannot be carried off by the stream is broken up with the hammer, or if too large, blasted with powder. Material not worth moving must be deposited where it is to remain, so that it will not be a source of expense.

This work must be carefully planned at the outset and systematically arranged, so that, if possible, no large rocks may be moved a second time. If the bank is covered with brush it should be cut off before the work commences. Trees must be cut down. It is better to remove the stumps at once with a stump puller, but if this is not done, the stumps, when washed out, must be either blasted or washed by the nozzle to a place where they can be piled up and remain permanently. To work economically on a large scale the openings should be so arranged as to work two or three, or perhaps more, nozzles in the same general direction. A single distributor can supply several nozzles which will work to within 180 ft. or 200 ft. of each other, according to the pressure of the water. At North Bloomfield the nozzle is 6 in. in diameter, with 240 ft. head. The feed pipe is 1200 ft. long and 40 in. in diameter. The pressure in the pipe is 108 lb. to the square inch, and at the nozzle 105 lb. to the square inch. The nozzle discharges 1300 miners' inches, or 4,212,000 cubic feet of water in twenty-four hours. Directly from the front of the work a sluice should lead away to the shaft. When the workings are very large, and several nozzles are being used at once, it will be necessary to cut a number of channels in the rock, or to construct sluices to carry the material towards the shaft in given directions, so that the whole of the ground shall not be inaccessible, but in some claims this is not necessary.

Any number of nozzles can be used on a bank provided they are not within about 200 ft. of each other. They may work independently or upon the same spot, as shown by the representation of the Malakoff Diggings in Figs. 25 and 26. The ground is worked in the shape of a funnel whose centre is the shaft connecting with the tunnel sluices, into which one or two surface sluices lead. In the subsequent workings the shaft may become entirely washed out, so that the whole working

will be one huge funnel with its opening at the head of the main tunnel.

Where the banks must be over 100 ft. high the working may become dangerous both to the men and the nozzles, for although the banks generally give ample warning before they cave, the rush of the caving dirt, from a height of over 100 ft., which has become very mobile from its mixture with so much water, may carry it a long distance into the mine, and instances have been known where the miners have been caught in the débris, from which it was scarcely possible to rescue them. To avoid such accidents high workings are usually divided into benches of 100 ft. or less, as at Sucker Flat. Not only the danger of accident is in this way avoided, but the opening of the mine is easier, for it gives an opportunity to be earning from the first bench the expenses which are necessary to open up the rest of the claim.

At the Polar Star Mine, Placer County, California, it was found advisable to work the deposit, which was on average 325 ft. thick, in two benches. Previous to 1880, the principal bench, which was 150 ft. high, had been piped off and the nozzles were directed upon the lower bench, which had a bank of over 100 ft. The necessity for working in benches does not, however, always arise. Banks sometimes as high as 250 ft. are worked in a single bench, though accidents to the apparatus and to the men are very apt to occur when deep deposits are attacked in this way, on account of the impossibility of getting out of the way of the run of the dirt. Generally beyond 250 ft. benches are used. In some parts of the Spring Valley Mine the banks are 450 ft. high, and when they were worked in a single bench it was not unusual for the caving to bury pipes which were throwing 7-in. streams, at a distance of 400 ft. from the face of the bank. If, for safety, the men and the nozzles are set very far back from the bank, the stream loses its force, and a large amount of power is thus wasted.

The following Table gives the average height of bank and yield of auriferous gravel in California* worked by the hydraulic method:

* Prod. Precious Metals in the U.S., 1881, p. 318.

Name of Company.	Average Height of Bank. ft.	Yield per Cubic Yard. cents.
Smartsville Claims, Yuba County	112	19½
Blue Tent, Nevada County	180	15
North Bloomfield, Nevada County	180 to 260	4 to 6½
Gold Run, Placer County	200	4½
Columbia Hill, Milton County	100	4½
La Grange, Stanislaus ,, ...	18 to 100	2½ to 15½
Patricksville, ,, ,, ...	40 to 60	4½ to 18½
Dardanelles, Placer County	150	13

The face of the bank must always be kept square, as work can be carried on most rapidly in this way at the least expense. If the banks are allowed to get concave, the men will very soon be surrounded by and under them, and will consequently be in a very dangerous position. If the banks should cave suddenly they would have but little chance of escape, and they would at all times be liable to be caught in an unexpected run. The higher the banks the greater the danger, and hence it is the practice of the most skilful miners to divide the banks into benches, so as to keep them within a minimum of between 125 ft. and 150 ft. I saw, however, at Smartsville, single benches over 200 ft. high, but such banks are very dangerous.

When a cave is about to occur it always gives warning; the water should then be turned off. The dirt will not run so far as to endanger the men if this is done, and only the boulders, which can be seen and avoided, will run. Arrangements should, therefore, always be made to cut the water off instantly from any part of the works. If the water was allowed to play upon the falling mass, the dirt would also run, and the men would have difficulty in avoiding it. Four-fifths of all the serious accidents and losses of life have occurred from want of care in this respect. It is generally best to effect caves before night, so as to have plenty of time to remedy any accident before the next day's work. In very large claims they work at night to clean up the cave, but, in small ones, a night shift is not usual unless the pay is very rich, and then the work is continued as much to prevent unlawful clean-ups as to get the pay in a shorter time.

When the sluices are new or recently set, it is necessary to

wash for half a day to a day with a full head of water on a light gravel, but without quicksilver, in order to fill the greater part of the interstices between the riffles and pavings, so that the mercury when added shall not sink so low that the gold will not reach it. After the openings are filled, the supply of water is diminished or entirely cut off, and from 500 lb. to 600 lb. of quicksilver added for a sluice 5000 ft. long. The upper parts are charged more liberally than the lower, as the quicksilver is always carried down stream. It is scattered from iron sprinkling pots with large holes made for the purpose, or poured from a vessel, the fingers being placed on the spout in such a way as to make several streams, and to break up the mercury as it falls into the sluice into small but not fine globules. It is not added all at once, but little by little during several days, until it is distinctly seen in the riffles and pavements. The amount of water used should always be sufficient to cover the largest stones that are sent down the sluice. If too much water is used with light gravel the sand is apt to pack. If the water is properly regulated, the rocks keep the sand moving and disintegrates the cement.

The quantity of quicksilver to be added varies, of course, with the richness of the dirt and the length of the sluices. The men judge from the consistency of the amalgam and the general appearance of the mercury in the sluices, which are constantly inspected, whether more mercury is needed. In some of the largest mines two or three tons of quicksilver are often in process of manipulation at one time, some of it being in the sluices and some of it in the shape of amalgam being cleaned or waiting for distillation.

In some claims it is the practice to diminish the quantity of mercury progressively during the last weeks of the run. There does not seem to be any good reason for this; the best practice is to carefully watch the sluices, and add, up to the last day, the mercury which seems to be necessary. The sluices are generally looked after once or twice a day during the run, and the mercury added as there is occasion for it. About 100 lb. per day is ordinarily used, so that 120 flasks of 74½ lb. each would

last for a three months' run. The quicksilver distilled from the amalgam collected in the upper part of the sluice boxes is always used at once.

The boulders which are found in washing, if small, are carried off through the main sluice, and are disposed of by the grizzly; large boulders are treated in different ways. A dangerous method which is used sometimes for very large ones, is to gradually undermine them, and allow them to roll. I saw at Smartsville a boulder over 6 ft. in diameter, which had been moved several thousand feet forward from a height of 250 ft. in this way. This is necessarily dangerous, though in most cases the direction in which the boulder is to roll can be easily secured. Those of very large size are sometimes moved by a derrick. This is of very simple construction; the mast is usually about 100 ft. high, the boom 80 ft. to 90 ft. long; the former is set in an iron sill and supported by guys. Once set up this derrick can be readily moved without being taken to pieces. It is erected only in places where large numbers of heavy boulders are likely to occur. Those weighing 11 tons have been moved by it; but if of larger size they must be blasted.

Hurdy-Gurdy Wheels.

The power which is employed to move the cranes, and often for stamps, diamond drills, and other purposes, is known as hurdy-gurdy wheels. They consist of two general types. In the first of these the rim of the wheel is made of wood, and to it brackets, 4 in. wide and 1 ft. apart, are attached. These brackets are of cast iron, and are very nearly square. The other type is a bucket wheel, which is made of wood or iron, having sides 5 in. in width, between which buckets of sheet iron are fastened 1 ft. apart and 4 in. deep at their opening, and slightly curved. The wheels vary in diameter from 3 ft. to 25 ft. The water is brought to them by means of pipes which strike the buckets at a tangent. These buckets are of two forms, single, like the Knight wheel, Fig. 32; or double, like the Pelton wheel, Fig. 33. The pressure of water varies from 250 ft. to 350 ft., and is brought by pipe 18 in. in diameter. They make from fifty to a hundred turns per minute. When they are small they are

sometimes placed horizontally; when they are large they are placed vertically in a box, the bottom of which is open to allow of the escape of the water. The older forms of these wheels are exceedingly simple in construction, but as they move very rapidly there is a great loss of power from the air which is carried in with them. They vary from 10 to 50 theoretical horse-power, but do not yield more than 30 to 40 per cent. Booth and Co., of San Francisco, have given the following Table of theoretical horse-power of the miners' inch for such wheels:

Heads in feet	100	90	80	70	60	50	40
	30	20	15	10	5	3	1
Inches to horse-power	3.25	3.61	4.05	4.64	5.41	6.50	8 12
	10.8	16.2	21.6	32.5	65.	108.	325.

These wheels have been improved by splitting the jet as it strikes a bifurcation on the double bucket, so that if necessary two nozzles can be used on the same wheel. Tests with this form of wheel show very great efficiency.*

Of the hurdy-gurdy wheels one of the best is the Knight's wheel, Fig. 32, which is extensively used. It is greatly superior to the styles formerly used both in the shape of the nozzles and the curve of the buckets, the ease with which the wheels are greased and the percentage of power which they utilise. The whole of the wheel is made of iron. The Table below gives the diameter and horse-power of the various sizes:

Diameter in Feet.	Horse-Power up to
1.6	2
1.6	5
2.6	7
2.6	10
3.0	10
3.0	20
3.0	40
4.6	30
4.6	40
4.6	60
6.0	40
6.0	60
6.0	80
8.0	125
8.0	200

* This subject has been ably discussed in bulletin No. 1 of the University of California on Hurdy-Gurdy Wheels, by R. E. Brown, 1883, and by Hamilton Smith; Trans. of Am. Soc. of Civil Engineers; vol. xiii., p. 21.

The nozzle is fixed in an iron box having a slit or aperture about ½ in. in width and 18 in. or 20 in. long, conforming to an arc of the circumference of the wheel. This peculiar form causes the discharged water to impinge directly upon several buckets at the same instant, thereby exerting an equal and uniform pressure upon the wheel; whereas, with round nozzles, no two areas of opening are at the same distance from the buckets, consequently the water discharged from the lower part of a round nozzle must pass through a space of from 4 in. to 8 in. before striking the buckets, thereby occasioning a considerable loss of power. In the "Knight" water-wheel the nozzle discharges within ⅛ in. from the buckets. The shape of the buckets is such that the delivery of water from the nozzle imparts both an impact and a reactionary power, and clears itself completely.

The Knight wheels are governed with great ease and uniformity, owing to the fact that the nozzle of the wheel is so constructed that perfect governing is accomplished by simply raising or lowering a steel tongue fitted in a groove at its outlet, by which the aperture of discharge is made longer or shorter, and the volume of water delivered thereby increased or decreased as may be required. This adjustment of the tongue neither changes the shape of the discharge outlet, nor does it interfere with the direction and impinging effect of the stream. This tongue is raised or lowered by means of a small hydraulic cylinder, which, in its turn, is supplied and exhausted from an hydraulic valve chest fixed immediately beneath, and under the control of the governor. This governor regulates the speed of the wheel so accurately that the least fractional amount of power can be controlled and transmitted from it. Where round nozzles are used, there are only two methods of government, *i.e.*, either to cut out a portion of the water as it leaves the nozzle, in which there is no economy, or to partially close the water gate, which virtually throttles the stream and necessarily wears the gate.

FIG 32. KNIGHT'S WHEEL.

The following Table gives powers of "hurdy-gurdy" wheels as developed by the "Knight" water-wheel:*

Head of Water.	Miners' Inches of Water.						Diameter of Wheels.					
	25	50	75	100	150	200	1.6	2.6	3.0	4.6	6.0	8.0
	Gives in Horse-Power						Revolutions per Minute.					
20	1.09	2.20	3.45	4.50	6.82	9.18	260	155	130	87	65	49
30	1.64	3.31	5.00	6.75	10.22	13.75	316	190	158	105	79	59
40	2.19	4.42	6.70	9.00	13.63	18.36	360	196	180	120	90	67
50	2.73	5.51	8.35	11.25	17.00	22.94	404	241	202	135	101	76
60	3.28	6.62	10.03	13.50	20.45	27.54	444	267	222	148	111	84
70	3.82	7.71	11.68	15.75	23.87	32.12	452	284	236	151	113	89
80	4.38	8.84	13.40	18.00	27.27	36.72	520	310	260	173	130	97
90	4.91	9.91	15.02	20.25	30.67	41.30	548	328	274	183	137	103
100	5.47	11.04	16.73	22.5	34.00	45.90	576	346	288	192	144	108
110	6.00	12.12	18.36	24.75	37.69	50.48	604	363	302	201	151	113
120	6.57	13.27	20.10	27.00	40.90	55.00	632	380	316	211	158	119
130	7.10	14.34	21.72	29.25	44.30	59.66	664	397	332	221	166	124
140	7.64	15.43	23.37	31.5	47.72	64.26	676	406	338	225	169	127
150	8.18	16.52	25.03	33.75	51.12	68.84	704	423	352	235	176	132
160	8.73	17.63	26.71	36.00	54.54	73.44	726	432	364	242	182	137
170	9.28	18.74	28.39	38.25	57.94	78.00	748	449	374	249	187	140
180	9.82	19.83	30.04	40.50	60.97	82.10	780	466	390	260	195	146
190	10.38	20.96	31.76	42.75	64.75	87.20	792	475	396	264	198	149
200	10.92	22.05	33.41	45.00	68.17	91.80	800	489	400	266	200	153
210	11.48	23.18	35.12	47.25	71.57	97.40	836	501	418	279	209	157
220	12.00	24.26	36.75	49.50	75.00	100.98	848	509	424	283	212	159
230	12.57	25.39	38.46	51.75	78.39	105.56	868	525	434	289	217	164
240	13.11	26.48	40.11	54.00	81.81	110.16	892	535	446	297	223	168
250	13.64	27.55	41.73	56.25	85.21	114.74	908	545	454	303	227	170
260	14.20	28.68	43.45	58.50	88.62	119.34	920	557	460	307	230	175
270	14.73	29.75	45.07	60.75	92.00	123.92	952	570	476	317	238	178
280	15.30	30.90	46.81	63.00	95.44	128.52	964	579	482	321	241	181
290	15.83	31.97	48.43	65.25	98.84	134.12	980	588	490	327	245	184
300	16.39	33.10	50.15	67.50	102.26	137.70	992	605	496	331	248	186
320	17.46	35.26	53.42	72.00	109.00	146.88	1028	627	514	343	357	194
340	18.55	37.47	56.76	76.50	115.89	156.00	1060	639	530	353	265	199
360	19.65	39.69	60.12	81.00	122.71	165.24	1096	656	548	365	274	205
380	20.74	41.49	63.46	85.50	129.53	174.42	1124	674	562	375	281	211
400	21.84	44.11	66.83	90.00	135.45	183.60	1152	691	576	385	288	216
425	23.21	46.88	71.00	95.62	144.86	195.00	1188	713	594	396	297	223
450	24.56	49.61	75.15	101.25	153.38	206.54	1224	735	612	408	306	229
475	25.93	52.47	79.34	106.87	161.90	218.00	1256	754	628	418	314	236
500	27.28	55.10	83.47	112.50	170.43	229.50	1288	774	644	429	322	242

Fig. 33 shows the Pelton wheel with its double buckets. The superiority of this wheel† was demonstrated in a com-

* Communicated by Joshua Hendy, Esq., of San Francisco.

† Communicated by L. A. Pelton, of Nevada City, California.

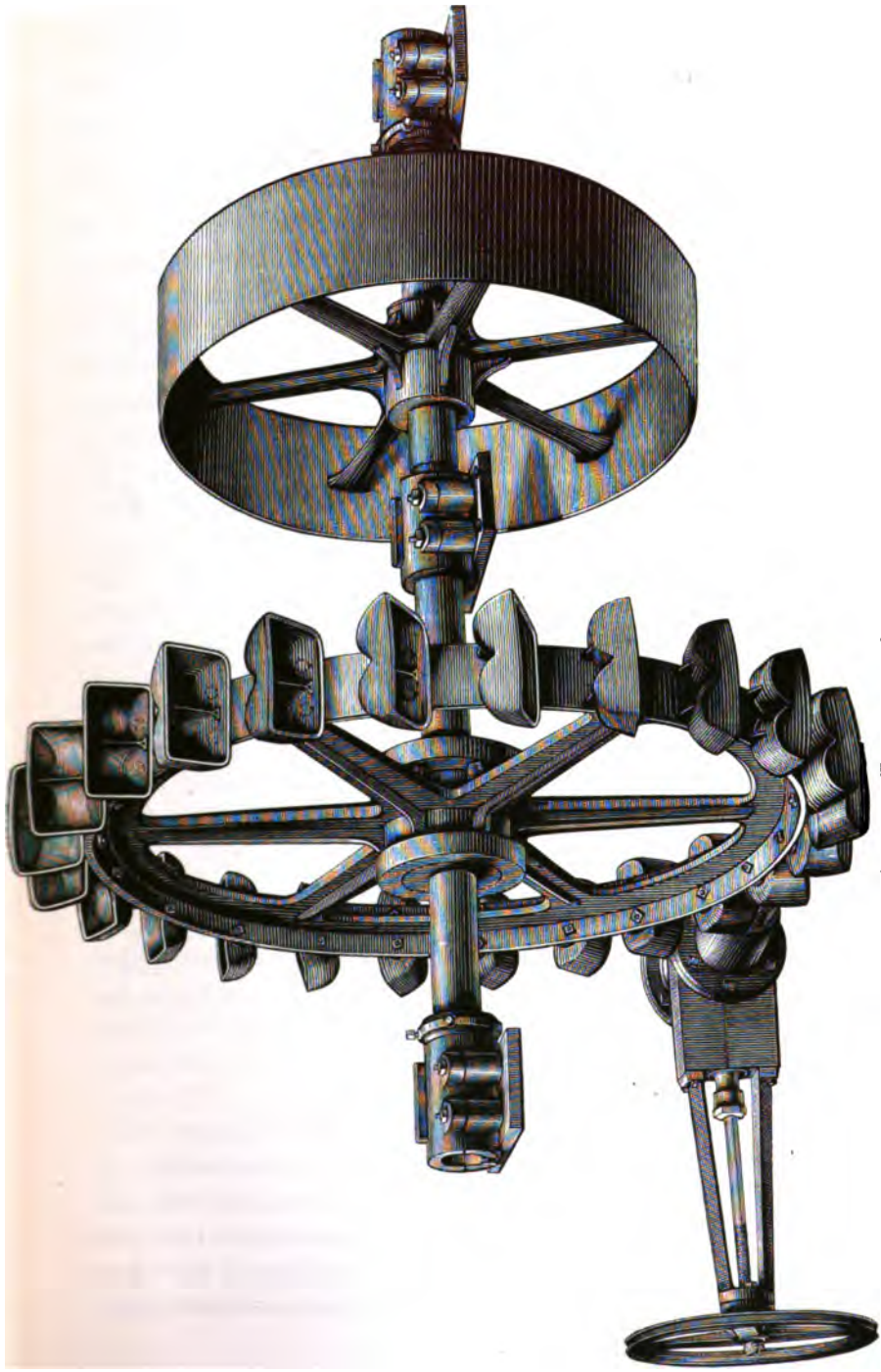


FIG. 33. THE PELTON 6-FT. WHEEL.

petitive trial between the Knight, Pelton, and two other pressure wheels, in a series of tests made at Grass Valley, Cal., by the Idaho Mining Company, for the purpose of ascertaining the percentage of efficiency of the various wheels in the market, with the view of adopting the best in driving their mining machinery. The tests were made with a Prony brake, the point of which, resting on the scales, described a circumference of 30 ft. The wheels were 6 ft. in diameter, and under a surveyed head of 386½ ft. The pressure gauge indicated 165 lb. standing, and 162 lb. running pressure. No allowance was made for loss of head by friction in the pipe. The water was measured over an iron weir ⅛ in. thick and 36½ in. long, without end contractions. Boyden's hook micrometer adjustment was used for finding depth over weir; its position was at one side, and 24 in. up stream from crest of weir, and indicated 4.975 in. depth when testing the Pelton wheel. The J. B. Francis formula for discharge over weirs was adopted as the basis of calculations. The tests showed that the Pelton wheel used 163.211 cubic feet of water per minute, and the average power developed in the numerous tests was 107.77 horse-power, being an average efficiency of 90.2 per cent. The Knight wheel gave an efficiency of 71.72 per cent. and the two other wheels 69.6 and 60.5 per cent. The Pelton wheel will keep its efficiency above 80 per cent. with great range in the quantities of water used, and can, with a constant head, always be kept up to 85 per cent.

When the wheel is to be set up the earth and rock should be tamped about the mud sills, and if great power is required, the mud sills should be bolted to a stone or wood foundation deeper in the ground. The wheel compartment is boarded up on the outside of the timbers, from the boxes down, a door being put in opposite the nozzle. The wheel is covered from 1 in. below the bottom of the boxes on the inside of the timbers. This cover is made loose, so it can be taken off, if occasion requires. The floor under the wheel is made of 1 in. lumber and double, to prevent leakage. The stream should be thrown parallel with the wheel, and be divided in its centre by the splitter in the centre of the buckets. These buckets are movable, and any of them can be replaced when necessary. To suit the power required, any

sized tip or nozzle may be screwed into the outer end of the standing nozzle. Variable or contracting nozzles that can be controlled by a governor are also made to go with them. In some cases wheels using two streams on a single wheel, each of which is under control of the operator, are used for hoisting and pumping.

The following Table gives capacity of each Pelton wheel under various heads:*

Diameter of Wheels in Feet.	25 ft. Head. HP.	50 ft. Head. HP.	75 ft. Head. HP.	100 ft. Head. HP.	200 ft. Head. HP.
1½	0.1 to 0.4	½ to 1	½ to 2	1 to 3	2 to 7
2	0.2 ,, 0.7	½ ,, 2	1 ,, 3	2 ,, 5	4 ,, 14
3	0.4 ,, 1.5	1 ,, 4	2 ,, 8	3 ,, 11	8 ,, 31
4	0.7 ,, 2.7	2 ,, 7	4 ,, 14	5 ,, 19	15 ,, 58
5	1.0 ,, 4.0	3 ,, 11	6 ,, 22	8 ,, 31	23 ,, 90
6	1.5 ,, 6.0	4 ,, 16	8 ,, 32	12 ,, 46	33 ,, 130
6	4.0 ,, 12.0	8 ,, 32	16 ,, 64	24 ,, 92	66 ,, 260

Diameter of Wheels in Feet.	300 ft. Head. HP.	400 ft. Head. HP.	500 ft. Head. HP.	600 ft. Head. HP.
1½	4 to 15	6 to 23	8 to 32	12 to 47
2	7 ,, 26	10 ,, 40	15 ,, 59	21 ,, 84
3	15 ,, 58	23 ,, 90	33 ,, 134	47 ,, 189
4	27 ,, 106	41 ,, 160	60 ,, 238	84 ,, 336
5	42 ,, 166	65 ,, 257	93 ,, 371	
6	61 ,, 240	93 ,, 370	134 ,, 535	
6	122 ,, 480	186 ,, 514		

As water under pressure can always be had in the vicinity of the great ditches, this application of it by means of hurdy-gurdy wheels is being extensively used where power is required, and has in many places replaced steam.

IV. The Clean-Up.

The time for making a clean-up is very variable and depends on a great variety of circumstances. As a general rule, the less often the sluice is disturbed the better. It will always have to run a little while without mercury at the commencement of a new run, and all such time and gravel is lost except in so far

* Communicated by L. A. Pelton, Nevada City, Nevada County, Cal.

as it prepares the sluice for the work which it has to do. The upper part of the sluice is cleaned oftener than the lower, for it is generally here that the largest quantity of gold is expected to be found. Some of the companies clean up the upper part of the sluice as often as twice a month, others only once a month, and others still once in six weeks or longer. When the water season is short the clean-up is made at the end of that time. A number of partial clean-ups are always made before a general one is undertaken. A general clean-up does not usually take place more than once or twice a year, when the whole working is stopped, and all the materials, such as pavements, false sides, and riffle boards taken from the sluices are placed in some convenient position on the outside so as to be used again.

When the clean-up is to be made the sluices are gradually emptied, down to the riffles and pavement, and these are removed, commencing with the upper ones. Riffles are then placed across the sluice, mercury is poured in to collect the amalgam, and the black sand and fine gravel are washed down with a small stream of water. The amalgam is scooped up and put into iron pails. Brooms, scrapers, knives, anything that will collect the amalgam, which may be caught in the crevices of the wood, or attached to any accidental projection of metal, are used. Very small particles of mercury or amalgam are often caught and firmly held in the pores or small holes or cracks in the wood. These are removed either with an old silver spoon, or a small piece of iron tipped with silver, to which it will adhere, when it could not otherwise be removed. The practised eye of the miner will readily detect these small particles which an ordinary observer would not notice.

At the Bloomfield Mine* the regular method is to clean up the sluice in the branch tunnel once in about two weeks, the length of the run depending upon the condition of the riffle blocks, and at the end of the season's run, usually just before the fall rains set in, to clean up all the places through which any gravel has been washed and where any gold is likely to have lodged; this latter constitutes the general clean-up of the mine. For a day or so before commencing the general clean-up, a head of clear water is run through the sluices to take out as much of the loose

* School of Mines Quarterly, vol. v., p. 374.

light sand and boulders as possible.* All the large rocks, chunks of pipe-clay, &c., are then blasted, and a second head of water is run through, which puts the mine in shape to be cleaned up with the least work. Starting at the lower end of the rock tunnel and at the same time at the lower end of the rock cut, lines of small sluice boxes, 12 in. wide and 11 in. deep, tapered at the ends to fit into each other, are set, and everything in the bottom of the tunnel is shovelled into these sluices and washed with a small head of water over the cross riffles with which they are provided. After the loose stuff is all shovelled in, the men work with iron scrapers, hooks, &c., to clean out every little crevice and hole in the rock; where the rock is soft and seamy they pick it up and throw it into the sluices, as the gold will work down to the bottom of the smallest hole. Whisk-brooms are used for sweeping out the cracks, and silver spoons for picking amalgam and quicksilver out of the holes. This method is carried out for the whole length of the tunnel, and the same operation is conducted in the rock cut leading to the sluice, the gold in the cut being, of course, bright gold, while nearly all found below the sluice will be amalgamated. The cleaning up in the cuts and sluices at the lower end of the mine is usually done a month or so before the work at the upper end is commenced, so as to have plenty of men; but the principle is precisely the same. After the tunnel and rock cut are thoroughly cleaned up, a head of water is turned on and run through the sluices, which works down any gold in the cut that may have remained, into the sluice. This is next cleaned up by taking out all the riffle blocks and allowing the amalgam and heavy material to fall on to the bottom. The head of water used during the operation carries off the lighter portion and leaves the amalgam dragging behind, which is scooped up and put into buckets; when the end of the sluice is reached and all the amalgam taken up, men go over the whole bottom with silver spoons and pick the amalgam and quicksilver out of every crack and nail hole. When this is finished, another head of water is run through which carries anything that may be left, down again through the tunnel to the under currents, tail cuts, &c. A little from the sluice is caught in the tunnel, but the amount is

* *Ibid.*, p. 375.

small, as the work in the sluice can be very thoroughly done. The tail cuts are cleaned up in the same way with the use of little sluice-boxes; the tail sluice, which is usually paved with rocks, so as to run a longer time, is cleaned up in the same manner as the main sluice, and the undercurrents are treated in the same way as the sluices. Opportunity is taken while the water is out of the mine and ditches, to put everything into thorough repair for the coming season. The quicksilver on hand which has been in use during the year is retorted, as it is found that some gold remains in it, after straining through drilling, the result being enough to pay well for the cost of retorting. By this time nearly the whole of the gold is collected and the mine put into shape for the next season's run.

AMALGAM OBTAINED FROM SECTIONS OF ROCK TUNNEL.*

Section.	Length in Feet.	Number of Ounces.	Value.
			$
Mouth up to No 1 shaft . .	1509	147	872.60
No. 1 shaft up to No. 2 . .	929.2	148.5	881.49
,, 2 ,, ,, 3 . .	906.5	222	1,317.87
,, 3 ,, ,, 4 . .	907	228	1,353.55
,, 4 ,, ,, 5 . .	889	410.5	2,436.97
,, 5 ,, ,, 6 . .	904.4	700.5	4,158.66
,, 6 ,, ,, 7 . .	962	2609.5	15,491.00
Total . . .	7007.1	4466	$26,512.14

SOURCES WHENCE GOLD WAS OBTAINED AT GENERAL CLEAN-UP OF 1883.

Source.	Ounces of Amalgam.	Value per Ounce.	Value.
		$	$
No. 10 cut	5,091.5	13.01	66,293.29
,, 10 flume	713	7.55	5,383.15
Tunnel	4,466	5.94	26,512.14
Tail cuts	356	6.01	2,142.25
,, sluice†	131.75	5.50	724.62
Undercurrents†	185	5.50	1,017.50
No. 8 cut	591	7.58	4,480.79
Total . . .	11,534.25	Av. $9.23	$106,553.74

* School of Mines Quarterly, vol. v., p. 377.

† These were cleaned up during the year. The amount from the tail sluice is the result of about one and a half months' run, and from the undercurrents is the result of a two months' and a half run.

The little sluice boxes,* unless in very good condition, are taken apart and burned and the ashes gathered up and panned out, yielding usually enough to pay for the price of the boxes. The tunnel is cleaned up in sections, a gang consisting of six or seven men starting with its line of boxes, say 120 ft. long, at the sump of one of the shafts and walking up to the next sump above, where gang No. 2 has already started. In this way the tunnel is cleaned up in about three or four days, the old and trusted hands taking the head section, which is, of course, the richest. The Table below shows the length of each section and the number of ounces of amalgam it produced. The gold obtained at the general clean-up of 1883, amounting to $106,553.74, is the result of a season's run of twelve months and twenty-four days, during which time 79,437 miners' inches of water were run at No. 8, and 865,296 in. were run at No. 10, Fig. 28, with a total yield, including the general clean-up, of $525,797.30, giving the percentage of gold for the general clean-up that year as $20\frac{3}{10}$ per cent. of the whole amount.

The amount of bullion produced during the last season up to the 1st of February, 1884,† at which time the mine was shut down by perpetual injunction from the United States Circuit Court, was $483,147.97. This amount was collected from the various sluices, undercurrents, &c., as follows:

	$
From No. 10 sluice	311,885.68
,, ,, 10 cut	104,146.19
,, ,, 8 sluice in tunnel	2,574.06
,, ,, 8 ,, mine	5,868.01
,, ,, 8 cut	2,129.39
,, undercurrents	13,194.08
,, tail sluices	3,708.75
,, cuts below the tunnel mouth	1,792.25
,, 7000 ft. of tunnel between sluice and undercurrents	21,749.14
Quicksilver, cleaning tanks, &c.	16,130.42
Total	$483,177.97

By far the largest part of the gold is caught in the first sluices, so that these admit of being much more frequently cleaned up than the other parts of the sluice. A very consider-

* School of Mines Quarterly, vol. v., p. 376.

† Prod. Gold and Silver in the U.S., 1883, p. 186.

able quantity of gold is caught in the undercurrents, so much so that, whatever the number of these undercurrents may be, it is sure that some gold will be caught in the last of them. It is soon calculated in each claim how often it is desirable to clean up each section, from the profit of doing it. This leaves the greater part of the sluice untouched during the larger part of the year, while the upper parts are cleaned up whenever it is desirable. When these partial clean-ups are being made, the work stands idle, so that double sets of sluices with gates, which converge into each other, or form, as at Smartsville, side switches in the main sluice, can be used with the greatest profit. There is probably no part of a sluice where gold and amalgam could not be found, but the question how to get it is the question which prevails when what is to be done is under consideration. Generally at least 80 per cent. of gold is caught in the first seventy yards at the head of the sluice; sometimes as much as 90 per cent. is found close to the head. The first undercurrent will generally catch about one-half of that caught by the undercurrents, and the second about one-third. These proportions will vary greatly, depending on the richness of the gravel, the size of the flume, and the velocity of the current. No matter how far removed the last undercurrent may be, something will be caught in it; the question where to stop depends on what it costs to catch the gold. If from heavy work or an unexpected cave, any of the upper part of the sluice is covered with gravel, the gold will be carried so much further down stream; but it will eventually be caught or lost in the tailings.

The result of 464 days' washing, using 607,833 twenty-four hour inches, yielded to the Yuba Gold Washing Company $395,420, which represented a washing of six acres. This was a yield of 65½ cents per cubic inch. The cost of production was $317,495, leaving a profit of $77,925.* The following Table† gives the results at the Gardners' Point claims for 1874, as follows:

	$
Gold collected in the first 150 ft. of the sluice ...	54,000
,, ,, other parts of the sluice ...	11,000
Amount collected in the undercurrents	3,000
Total amount collected in 100 days' run...	$68,000

* Prod. Gold and Silver in the U.S., 1881, p. 630.

† Trans. Am. Inst. Min. Eng., vol. vi., p. 50.

	$
Yield of first undercurrent :	
796 ft. from head of sluice	1500
Yield of 2nd undercurrent :	
874 ft. from head of sluice	1000
Yield of 3rd undercurrent :	
965 ft. from head of sluice	500
Total yield of undercurrents	$3000

As a protection against thieves, the sluices, which are worked at intervals, are either filled with gravel before turning off the water, and the gravel removed by hand from those sections only which are being cleaned, or a stream of water is kept running through them to prevent others from cleaning up. This last is an expensive method except when the claim owns the water supply. In some cases, as at Smartsville, the sluices when the water is turned off are protected by armed sentinels.

The quantity of gold contained in the gravel can only be approximated, as it can only be estimated after a long run and clean-up. It was found after a careful examination of the upper gravel in the district of Gold River, Placer County, that the yield was about $0.0475 per cubic yard. At French Corral, Nevada County, the average yield of the upper ground was 15 cents, and of the cement and lower strata $3.50. It was found that the cost of working the cement was $1.10, or a clear profit of $2.40. At the Spring Valley Mine the most valuable part of the gravel yielded from $1 to $8, but this is exceptionally rich and only true of the richest part.

V. Treatment of the Gold.

The amalgam which is collected is never pure. It contains beside gold some dirt and sand mechanically mixed with it, and a certain quantity of base metal, from which it must be freed. It is first broken into small pieces and washed with water, which is sometimes heated with steam to free the amalgam from dirt. A large excess of fresh mercury is then added, thoroughly mixed with it, and allowed to settle. The amalgam containing the gold will for the most part go to the bottom, and that containing the base metals rise to the top. The upper part is carefully removed, and the lower part again washed, and the excess of mercury

removed by straining through a conical canvas bag,* or when the amount is small, through leather. It is then treated with a mixture of one-third sulphuric acid and two-thirds water, and heated over a slow fire from half to three-fourths of an hour.

The skimmings are treated with nitric acid to remove the base metals, washed carefully with water, strained, and the amalgam added to the other. This amalgam is poorer than the first, but there is generally not enough of it to make it worth while to keep it separate. The amalgam is now ready to be retorted. This is done in a covered conical cast-iron vessel, Fig. 34, 8 in. in diameter above, and 6 in. below, and 12 in. deep, provided with

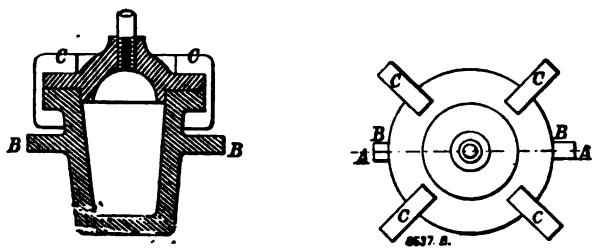


FIG. 34. GOLD AMALGAM RETORT.

trunnions on the outside so as to swing over the fire. This vessel is first carefully coated with a thin wash of clay and water to prevent the amalgam from sticking. The amalgam is very irregular in size, and is generally placed in the pot in the shape of small balls. The cover, which has a wrought-iron pipe cast in it, is fitted after the pot is filled. The cover is held to the pot by clamps, and is carefully luted. A condenser is adjusted to the cover pipe, and the vessel swung over a very hot fire and the distilled quicksilver collected under water; twenty minutes after the mercury ceases to flow the fire is drawn, and after the pot is cool enough the gold is taken out. It is of very variable fineness, depending of course on the quality of the gold in the claim.

As all the buildings are of wood, and consequently likely to burn, and as there is always a desire more or less strong to clean

* See vol. i., p. 395.

up other people's property, the supply of mercury is, at Smartsville, buried in the ground.

In most of the large mines the retorted gold is broken, placed in a crucible with borax, and fused in a hot charcoal fire. It is then cast in a mould into the shape of a brick, hence the name "gold bricks." After cooling, the brick is stamped with the name of the company and the weight of the bar, and sent to be assayed. The assay value and the authority for it is stamped upon it, and it then finds its way into the market.

At most of the small mines the retorted amalgam is usually sold in the crude state. As we have already seen, the bullion from the placers is very much finer than that obtained from any other source. The alloy is rarely anything else than silver.

The following Tables give the fineness of placer gold in some of the States and Territories of the United States.* No account is taken by the miners of the silver alloyed with placer gold, unless in the exceptional cases where the value of the former is allowed for in the sale of the product, as in direct sales to the mints or United States assay offices. In the aggregate this silver forms a considerable item.

EXAMPLES OF THE FINENESS OF PLACER GOLD.

CALIFORNIA.

County and Locality.	Fineness in Gold.	Remarks.
BUTTE.		
Centerville	0.900	
Cherokee Flat	0.970	
Magalia	0.900	
Morris Ravine	0.905—0.908	
Oroville	0.942	
CALAVERAS.		
Mokelumne Hill	0.850—0.960	
DEL NORTE.		
Bunker Hill Mine	0.887—0.925	
China Creek	0.900	
Happy Camp	0.875—0.935	
Indian Creek	0.900	
Wingate Mine	0.944—0.950	
EL DORADO.		
Placerville	0.980	

* Report of U.S. Census, 1880, vol. xiii., p. 350.

CALIFORNIA—*continued.*

County and Locality.	Fineness in Gold.	Remarks.
HUMBOLDT.		
Gold Bluffs	0.940	
Orleans Bar	0.726	Sold for $13 to $17 per ounce.
PLACER.		
Bath	0.850	
Dutch Flat	0.900—0.910	
Gold Run	0.900	
Iowa Hill	0.784—0.814	
Michigan Bluffs	0.925—0.960	
PLUMAS.		
Black Hawk Mine	0.924	
Cook's Cañon	0.885	
Gopher Hill	0.936	
Hungarian Hill	0.924	
Seneca	0.846	
SHASTA.		
Igo	0.885	
SISKIYOU.		
Callahan's Ranch	0.859	Sold for $17.75 per ounce.
Cottonwood	0.860	
Coyote Gulch	0.900—0.950	
Galena Hill	0.880—0.930	
Greenhorn	0.850	
Humbug Creek	0.809—0.865	
Indian Creek	0.835—0.900	Sold for $17 per ounce.
McAdam's Creek	0.750—0.900	
Oro Fino	0.762	
Rattlesnake Creek	0.827	Sold for $16.50 per ounce.
Sawyer's Bar	0.850	
Sciad Valley	0.887—0.912	
Scott ,,	0.805	Sold for $16 to $16.25 per ounce.
South Fork Salmon	0.835	Sold for $16.75 per ounce.
Yreka	0.749—0.863	Sold for $15 to $17.12 per ounce.
STANISLAUS.		
La Grange	0.920	
TRINITY.		
Buckeye	0.890—0.900	
Cañon Creek	0.894—0.896	
Chapman's and Fisher's Mines	0.914—0.917	
Douglas	0.912	
Indian Creek	0.920	
Junction City	0.875—0.915	Sold for $18 per ounce.
Red Hill	0.910—0.917	
Trinity Center	0.900	
,, Mine	0.887—0.903	
Weaver	0.906—0.927	

COLORADO.

County and Locality.	Fineness in Gold.	Remarks.
CHAFFE.		
Hope	0.850	
CLEAR CREEK.		
Jackson	0.880	
LAKE.		
California	0.850—0.875	
PARK.		
Alma Township	0.819	
ROUTT.		
Hahn's Peak	0.666	
SUMMIT.		
Bevan	0.750—0.820	
Spalding	0.850—0.900	

DAKOTA.

LAWRENCE.		
Bear Gulch	0.940	
Cape Horn	0.870	
PENNINGTON.		
Cañon	0.925	
Confederate	0.925—0.940	
Jenny and Strawberry	0.940	

GEORGIA.

CHEROKEE.		
Fifteenth District	0.927	Sold for $19.20 per ounce.
HALL.		
810 G.M. 9th and 12th districts.	0.900—9.916	
LUMPKIN.		
Twelfth District	0.875—0.995	
Thirteenth District	0.962—0.992	
WHITE.		
Fourth District	0.945	
UNION.		
Tenth District	0.981	

IDAHO.

BOISE.		
Boisé Basin and vicinity	0.7806	Average of 413 lots.

MONTANA.

County and Locality.	Fineness in Gold.	Remarks.
BEAVER HEAD.		
Bannock	0.935	
DEER LODGE.		
Gold Hill	0.950—0.956	
Henderson Gulch	0.925—0.957	
Independence	0.725—0.730	
McClellan's Gulch	0.860—0.930	Sold for $17.50 per ounce.
Nelson	0.935—0.940	
Summit Valley	0.700	
LEWIS AND CLARKE.		
Last Chance	0.910	
MEAGHER.		
German	0.830—0.835	
Thompson's Gulch	0.945—0.980	

OREGON.

BAKER.		
Amelia	0.810—0.900	
Blue Cañon	0.840—0.853	Sold for $16.50 per ounce.
Chicken Creek	0.725	Sold for $15 per ounce.
Humboldt Basin	0.789—0.866	,, ,,
Mormon Basin	0.750	,, $16 ,,
Pocahontas	0.782—0.851	,, $15.50 to $17 per ounce.
Rye Valley	0.756	
Shasta	0.850—0.860	Sold for $16.50 per ounce.
Willow Creek	0.825	
CURRY.		
Sixes River	0.825	Sold for $17 per ounce.
DOUGLAS.		
Big Bend	0.925—0.950	Sold for $17 per ounce.
Cañonville	0.900—0.956	
Green Mountain	0.825—0.830	
GRANT.		
Cañon City	0.850—0.900	Sold for $17.60 per ounce.
Granite	0.750—0.761	
Marysville	0.805—0.887	Sold for $16.25 to $17.60 per ounce.
Rock Creek	0.857	Sold for $16.50 to $16.75 per ounce.
Trail ,,	0.850	
JACKSON.		
Applegate	0.850	
Ashland	0.846	
Coyote Creek	0.895—0.930	
Dry Diggings	0.975	

OREGON—*continued.*

County and Locality.	Fineness in Gold.	Remarks.
JACKSON—*continued.*		
Fort Lane	0.870	Sold for $16.25 per ounce.
Forty-Nine	0.837	Sold for $16.50 per ounce.
Grass Creek	0.875—0.900	
Jackass Creek	0.950	
Jacksonville	0.860	
Sam's Valley	0.825	
Sardine Creek	0.810—0.900	Sold for $16.75 per ounce.
Sterling	0.826—0.908	
Uniontown	0.804 —0.820	Sold for $16.75 per ounce.
Wolf Creek	0.925	
JOSEPHINE.		
Althouse	0.865—0.875	Sold for $16.50 per ounce.
Cañon Creek	0.900	Sold for $17 per ounce.
Grass ,,	0.909—0.930	
Illinois	0.900	
Josephine	0.900	
Murphy	0.900	
Silver Creek	0.900	
Waldo	0.910—0.927	
Yank	0.967	
UMATILLA.		
Columbia River	0.740	Sold for $15.30 per ounce.
WASCO.		
Ochoco	0.711	

The following condensed abstract* shows the average fineness in gold and silver of the placer yield of several States and Territories, with the number of specimen examples from which these averges are derived. The amount of base metal contained in the crude bullion varies from nothing to 0.020.

AVERAGE FINENESS OF PLACER GOLD.

State or Territory.	Number of Examples.	Average Fineness.	
		Gold.	Silver.
California	80	0.8836	0.1124
Colorado	9	0.8205	0.1755
Dakota	7	0.9235	0.0725
Georgia	10	0.9228	0.0732
Idaho	413	0.7806	0.2134
Montana	14	0.8951	0.1009
Oregon	77	0.8727	0.1233
Total	610		

* Report of U.S. Census, 1880, vol. xiii., p. 352.

From eighty examples of California placer dust and bullion an average fineness of 0.8836 is derived. It is possible that this average is a trifle too high, owing to the natural tendency of producers to over-estimate the fineness of their gold.

The average for Idaho is of 413 lots of placer gold from Boisé Basin, a district producing three-fourths of the total for the territory. This gold is of less fineness than that obtained in several other localities in Idaho, but the average stated will hold as a close approximation for the total. The following Table gives the method of ascertaining the average fineness of gold found in the United States:*

State or Territory.	Average Fineness in Gold.		Yield of Placer Gold in Millions of Dollars.		Resultants.
California	0.8836	×	8.581	=	7.5821716
Colorado...	0.8205	×	0.102	=	0.0836910
Dakota	0.9235	×	0.051	=	0.0470985
Georgia	0.9228	×	0.067	=	0.0618276
Idaho	0.7806	×	0.880	=	0.6869280
Montana...	0.8951	×	1.163	=	1.0410013
Oregon	0.8727	×	0.926	=	0.8081202
			11.770		10.3108382

$$\text{Total average gold fineness} = \frac{10.3108382}{11.770} = 0.87602$$

It is found that the 580,766.6 oz. of fine gold were associated with 80,177.3 oz. of fine silver and 2753.2 oz. of base metal, making the total weight of the crude placer gold 663,697.1 oz. In mint value the silver contents were worth only $103,661, as against $12,005,511 for the gold. Thus while the ratio of silver to gold products was 1 : 7.2437 in weight, it was only 1 : 115.8153 in assay value. At the market rates the value of this placer silver, if any account were taken of it, would be considerably less, or a ratio of about 1 : 131. The loss to the miners in the price paid for the actual gold contents of the placer dust, when sold in small quantities to local dealers, is so much greater than the whole value of the silver contents, that the latter does not appear to them a matter of any consequence. The amount of silver alloyed with the placer gold of the several States and Territories is shown in the accompanying Table:†

* *Ibid.*, p. 353. † *Ibid.*, p. 358.

COMPOSITION AND MINT VALUE OF CRUDE PLACER GOLD.

State or Territory.	Contents of Placer Gold.				Mint Value.		
	Fine Gold.	Fine Silver.	Base Metal.	Total Crude Metal.	Gold.	Silver.	Total.
	oz.	oz.	oz.	oz.	$	$	$
Alaska	287.9	*39.4	1.3	328.6	5,951	*51	6,002
Arizona	*1,451.2	*198.8	*6.6	*1,656.6	*29,999	*257	*30,256
California	415,105.0	52,804.2	1,879.1	469,788.3	8,580,982	68,271	8,649,253
Colorado	4,921.9	1,052.8	24.0	5,998.6	101,745	1,361	103,106
Dakota	2,460.3	193.1	0.2	2,653.6	50,859	250	51,109
Georgia	3,234.5	256.6	14.0	3,505.1	66,863	332	67,195
Idaho	42,552.8	11,633.1	327.1	54,512.9	879,644	15,040	894,684
Montana	56,255.6	6,341.4	251.4	62,848.4	1,162,906	8,199	1,171,105
Nevada	*2,418.7	*331.3	*11.0	*2,761.1	*49,999	*428	*50,427
North Carolina	*226.7	*31.0	*1.0	*258.8	*4,686	40	4,726
Oregon	44,811.5	6,331.2	205.4	51,348.1	926,336	8,186	934,522
South Carolina	316.4	*43.3	*1.4	*361.2	6,541	*56	*6,597
Utah	967.5	*132.5	*4.4	*1,104.4	20,000	*171	*20,171
Washington	5,756.6	*788.6	*26 3	*6,571.4	119,000	*1,019	*120,019
Total	580,766.6	80,177.3	2,753.2	663,697.1	$12,005,511	$103,661	$12,109,172

* Estimated.

The following Tables give the production of both gold and silver in the various States and Territories of the United States in 1880:*

PACIFIC DIVISION.

State or Territory.	Product.		
	Gold.	Silver.	Total.
	$	$	$
Alaska	5,951	51	6,002
Arizona	211,965	2,325,825	2,537,790
California	17,150,941	1,150,887	18,301,828
Idaho	1,479,653	464,550	1,944,203
Nevada	4,888,242	12,430,667	17,318,909
Oregon	1,097,701	27,793	1,125,494
Utah	291,587	4,743,087	5,034,674
Washington	135,800	1,019	136,819
Total	$25,261,840	$21,143,879	$46,405,719

DIVISION OF THE ROCKY MOUNTAINS.

Colorado	2,699,898	16,549,274	19,249,172
Dakota	3,305,843	70,813	3,376,656
Montana	1,805,767	2,905,068	4,710,835
New Mexico	49,354	392,337	441,691
Wyoming	17,321	...	17,321
Total	$7,878,183	$19,917,492	$27,795,675

EASTERN DIVISION.

Alabama	1,301	...	1,301
Georgia	81,029	332	81,361
Maine	2,999	7,200	10,199
Michigan	...	25,858	25,858
New Hampshire	10,999	16,000	26,999
North Carolina	118,953	140	119,093
South ,,	13,040	56	13,096
Tennessee	1,998	...	1,998
Virginia	9,321	...	9,321
Total	$239,640	$49,586	$289,226

SUMMARY.

Pacific division	25,261,840	21,143,879	46,405,719
Division of the Rocky Mountains	7,878,183	19,917,492	27,795,675
Eastern division	239,640	49,586	289,226
Total	$33,379,663	$41,110,957	$74,490,620

* Report of U.S. Census, 1880, vol. xiii., p. 358.

PERCENTAGE OF PLACER GOLD AND GOLD FROM DEEP MINES IN THE TOTAL GOLD PRODUCT OF 1880.*

State or Territory.	Hydraulic, Placer, Drift, and River Mines.	Deep Mines.
Alabama	...	100.00
Alaska	100.00	
Arizona	14.15	85.85
California	50.03	49.97
Colorado	3.77	96.23
Dakota	1.54	98.46
Georgia	82.52	17.48
Idaho	59.45	40.55
Maine	...	100.00
Montana	64.40	35.60
Nevada	1.02	98.98
New Hampshire	...	100.00
,, Mexico	...	100.00
North Carolina	3.94	96.06
Oregon	84.39	15.61
South Carolina	50.16	49.84
Tennessee	...	100.00
Utah	6.86	93.14
Virginia	...	100.00
Washington	87.63	12.37
Wyoming	...	100.00
United States (including Alaska)	36.06	64.03

RELATIVE PRODUCTION OF THE STATES AND TERRITORIES IN 1880.

State or Territory.	Gold.			Silver.	Total.
	Placer Mines.	Deep Mines.	Total Gold.		
	per cent.	per cent.	per cent.	per cent.	per cent.
Alabama	...	0.01	0.01		
Alaska	0.05	...	0.02		
Arizona	0.25	0.85	0.63	5.66	3.40
California	71.47	40.09	51.38	2.80	4.57
Colorado	0.85	12.16	8.09	40.25	5.84
Dakota	0.42	15.23	9.90	0.17	4.53
Georgia	0.56	0.07	0.24	...	0.11
Idaho	7.33	2.81	4.43	1.13	2.61
Maine	...	0.01	0.01	0.02	0.01
Michigan	...	...	...	0.06	0.03
Montana	9.69	3.01	5.41	7.07	6.34
Nevada	0.42	22.64	14.64	30.24	23.27
New Hampshire	...	0.05	0.03	0.04	0.04
,, Mexico	...	0.23	0.15	0.95	0.59
North Carolina	0.04	0.53	0.36	...	0.16
Oregon	7.71	0.80	3.29	0.07	1.51
South Carolina	0.05	0.03	0.04	...	0.02
Tennessee	...	0.01	0.01		
Utah	0.17	1.27	0.87	11.54	6.76
Virginia	...	0.04	0.03	...	0.01
Washington	0.99	0.08	0.41	...	0.18
Wyoming	...	0.08	0.05	...	0.02
Total	100.00	100.00	100.00	100.00	100.00

* *Ibid.*, p. 359.

The relative proportion of placer gold to that produced by the deep mines, the percentage in each class of product of the precious metals from the several mining regions of the United States, are given in the preceding pages; the average yield per square mile and per capita in different localities, and the rank of the States and Territories in productiveness, are given in the Tables on pages 265 to 269.*

PRODUCTION UNACCOUNTED FOR IN THE PRECEDING TABLES.

In addition to the returns received directly from the mines, there are several minor points to be included in the total yield. A larger item than it is usually considered to be is the annual hoarding of rich specimens. This is not accounted for in the mine production as reported. While it is impossible to state the actual amount thus absorbed with any degree of precision, a careful estimate would place the value of the gold nuggets and ore annually added to the cabinets of collectors at not less than $150,000, and that of the silver ore at about $50,000. This, in view of the great number of mineral collections maintained throughout the mining territory, is certainly not an overestimate.

There is also quite an extensive manufacture of gold quartz into jewellery and souvenirs, particularly in San Francisco. The value so absorbed probably does not fall short of $50,000 annually. In 1870 the United States Mining Commissioner estimated the amount of gold hoarded as specimens or worked up by local jewellers at $400,000. The same authority, at that period, estimated the annual loss of gold dust in handling as currency at $100,000. As the practice of using dust for money has almost disappeared, the amount so lost is now very small.

Another indefinite quantity is the value of precious metal lost in melting, in assay grains, &c. Summing up the estimates for these additional items, the following result is reached:

PRODUCTION UNACCOUNTED FOR IN THE TABLES.†

	Gold.	Silver.	Total.
	dols.	dols.	dols.
Bullion product shown in preceding Tables	33,379,643	41,110,957	74,490,620
Estimated value of specimens hoarded	150,000	50,000	200,000
" " gold quartz made into jewellery and souvenirs	50,000	..	50,000
" " " dust lost in handling as currency	10,000	..	10,000
" loss in melting and assaying, assay grains, &c.	20,000	10,000	30,000
Total	33,609,643	41,170,957	74,780,620

* *Ibid.*, p. 361. † *Ibid.*, p. 359.

AVERAGE PRODUCT OF GOLD AND SILVER PER SQUARE MILE AND PER CAPITA IN 1880.*

States and Territories.	Average Product.					
	Per Square Mile.			Per Capita.		
	Gold.	Silver.	Total.	Gold.	Silver.	Total.
	$	$	$	$	$	$
Alabama	0.02	...	0.02	0.001	...	0.001
Alaska	0.01	...	0.01	0.20	...	0.20
Arizona	1.87	20.58	22.45	5.24	57.51	62.75
California	108.30	7.27	115.57	19.83	1.33	21.16
Colorado	25.98	159.24	185.22	13.89	85.16	99.05
Dakota	22.17	0.48	22.65	24.46	0.52	24.98
Georgia	1.36	0.01	1.37	0.05	0.0002	0.05
Idaho	17.45	5.48	22.93	45.37	14.25	59.62
Maine	0.09	0.22	0.31	0.005	0.011	0.016
Michigan	...	0.44	0.44	...	0.02	0.02
Montana	12.36	19.89	32.25	46.11	74.19	120.30
Nevada	44.16	112.29	156.45	78.51	199.63	278.14
New Hampshire	1.18	1.72	2.90	0.03	0.05	0.08
New Mexico	0.40	3.20	3.60	0.41	3.28	3.69
North Carolina	2.28	0.003	2.28	0.08	0.0001	0.08
Oregon	11.43	0.29	11.72	6.28	0.16	6.44
South Carolina	0.43	...	0.43	0.01	...	0.01
Tennessee	0.05	...	0.05	0.001	...	0.001
Utah	3.43	55.82	59.25	2.03	32.94	34.97
Virginia	0.22	...	0.22	0.006	...	0.006
Washington	1.96	0.015	1.98	1.81	...	1.81
Wyoming	0.18	...	0.18	0.83	...	0.83
United States (including Alaska)	9.31	11.44	20.75	0.66	0.82	1.48
,, ,, (not including Alaska)	11.03	13.59	24.62	0.66	0.82	1.48
,, ,, (including only the States and Territories producing gold and silver, with Alaska)	14.68	18.02	32.70	2.60	3.20	5.80
United States (including only the States and Territories producing gold and silver, and not including Alaska)	19.44	23.94	43.38	2.61	3.21	5.82
Average for Colorado, California, Nevada, Utah, Montana, Dakota, Arizona, and Idaho	33.47	42.74	76.21	21.04	26.87	47.91

* *Ibid.*, p. 360.

RANK OF THE STATES AND TERRITORIES IN ACTUAL PRODUCTION OF THE PRECIOUS METALS FROM 1880 TO 1886.

1880.			1881.		
Gold.	Silver.	Total.	Gold.	Silver.	Total.
1. California.	1. Colorado.	1. Colorado.	1. California.	1. Colorado.	1. Colorado.
2. Nevada.	2. Nevada.	2. California.	2. Dakota.	2. Arizona.	2. California.
3. Dakota.	3. Utah.	3. Nevada.	3. Colorado.	3. Nevada.	3. Nevada.
4. Colorado.	4. Montana.	4. Utah.	4. Montana.	4. Utah.	4. Arizona.
5. Montana.	5. Arizona.	5. Montana.	5. Nevada.	5. Montana.	5. Utah.
6. Idaho.	6. California.	6. Dakota.	6. Idaho.	6. Idaho.	6. Montana.
7. Oregon.	7. Idaho.	7. Arizona.	7. Arizona.	7. California.	7. Idaho.
8. Utah.	8. New Mex.	8. Idaho.	8. Oregon.	8. New Mex.	8. Oregon.
9. Arizona.	9. Dakota.	9. Oregon.	9. New Mex.	9. Dakota.	9. New Mex.
10. Washington.	10. Michigan.	10. New Mex.	10. Utah.	10. Oregon.	10. Washington.
11. N. Carolina.	11. Oregon.	11. Washington.	11. Washington.	11. Maine.	11. Georgia.
12. Georgia.	12. N. Hampshire.	12. N. Carolina.	12. Georgia.		12. N. Carolina.
13. New Mex.	13. Maine.	13. Georgia.	13. N. Carolina.		13. S. Carolina.
14. Wyoming.		14. N. Hampshire.	14. S. Carolina.		14. Alaska.
15. S. Carolina.		15. Michigan.	15. Alaska.		15. Virginia.
16. N. Hampshire.		16. Wyoming.	16. Virginia.		17. Wyoming.
17. Virginia.		17. S. Carolina.	17. Wyoming.		18. Tennessee.
18. Alaska.		18. Maine.	18. Tennessee.		
19. Maine.		19. Virginia.			
20. Tennessee.		20. Alaska.			
21. Alabama.		21. Tennessee.			
		22. Alabama.			

1882.			1883.		
Gold.	Silver.	Total.	Gold.	Silver.	Total.
1. California.	1. Colorado.	1. Colorado.	1. California.	1. Colorado.	1. Colorado.
2. Colorado.	2. Arizona.	2. California.	2. Colorado.	2. Montana.	2. California.
3. Dakota.	3. Utah.	3. Nevada.	3. Dakota.	3. Utah.	3. Nevada.
4. Montana.	4. Nevada.	4. Arizona.	4. Nevada.	4. Nevada.	4. Montana.
5. Nevada.	5. Montana.	5. Utah.	5. Montana.	5. Arizona.	5. Arizona.
6. Idaho.	6. Idaho.	6. Montana.	6. Idaho.	6. New Mex.	6. Utah.
7. Arizona.	7. New Mex.	7. Idaho.	7. Arizona.	7. Idaho.	7. Idaho.
8. Oregon.	8. California.	8. Dakota.	8. Oregon.	8. California.	8. Dakota.
9. Georgia.	9. Dakota.	9. New Mex.	9. Alaska.	9. Dakota.	9. New Mex.
10. Utah.	10. Oregon.	10. Oregon.	10. New Mex.	10. Oregon.	10. Oregon.
11. New Mex.	11. N. Carolina.	11. Georgia.	11. Georgia.	11. N. Carolina.	11. Alaska.
12. N. Carolina.		12. N. Carolina.	12. N. Carolina.	12. Georgia.	12. Georgia.
13. Alaska.		13. Alaska.	13. Utah.	13. Washington.	13. N. Carolina.
14. Washington.		14. Washington.	14. Washington.	14. S. Carolina.	14. Washington.
15. S. Carolina.		15. S. Carolina.	15. S. Carolina.		15. S. Carolina.
16. Virginia.		16. Virginia.	17. Virginia.		17. Virginia.
17. Wyoming.		17. Wyoming.	18. Wyoming.		18. Wyoming.

1884.			1885.		
Gold.	Silver.	Total.	Gold.	Silver.	Total.
1. California.	1. Colorado.	1. Colorado.	1. California.	1. Colorado.	1. Colorado.
2. Colorado.	2. Montano.	2. California.	2. Colorado.	2. Montana.	2. California.
3. Nevada.	3. Utah.	3. Montana.	3. Montana.	3. Utah.	3. Montana.
4. Dakota.	4. Nevada.	4. Nevada.	4. Dakota.	4. Nevada.	4. Nevada.
5. Montana.	5. Arizona.	5. Utah.	5. Nevada.	5. Arizona.	5. Utah.
6. Idaho.	6. California.	6. Arizona.	6. Idaho.	6. Idaho.	6. Idaho.
7. Arizona.	7. New Mex.	7. Idaho.	7. Arizona.	7. New Mex.	7. Arizona.
8. Oregon.	8. Idaho.	8. Dakota.	8. New Mex.	8. California.	8. New Mex.
9. New Mex.	9. Dakota.	9. New Mex.	9. Oregon.	9. Dakota.	9. Dakota.
10. Alaska.	10. Oregon.	10. Oregon.	10. Alaska.	10. Washington.	10. Oregon.
11. S. Carolina.	11. Other States.	11. Alaska.	11. Utah.	11. Oregon.	11. Alaska.
12. Georgia.	12. N. Carolina.	12. N. Carolina.	12. N. Carolina.	12. Other States.	12. Washington.
13. Utah.	13. Washington.	13. Georgia.	13. Georgia.	13. N. Carolina.	13. N. Carolina.
14. Washington.	14. S. Carolina.	14. Washington.	14. Washington.	14. Alaska.	14. Georgia.
15. Other States.		15. Other States.	15. Other States.	15. Wyoming.	15. Other States.
16. S. Carolina.		16. S. Carolina.	16. S. Carolina.		16. S. Carolina.
17. Wyoming.		17. Wyoming.	17. Wyoming.		17. Wyoming.
18. Virginia.		18. Virginia.	18. Virginia.		18. Virginia.

1886.		
Gold.	Silver.	Total.
1. California.	1. Colorado.	1. Colorado.
2. Colorado.	2. Montana.	2. Montana.
3. Montana.	3. Utah.	3. California.
4. Nevada.	4. Nevada.	4. Nevada.
5. Dakota.	5. Idaho.	5. Utah.
6. Idaho.	6. Arizona.	6. Idaho.
7. Arizona.	7. New Mex.	7. Arizona.
8. Oregon.	8. California.	8. Dakota.
9. Alaska.	9. Dakota.	9. New Mex.
10. New Mex.	10. Other States.	10. Oregon.
11. Utah.	11. Washington.	11. Alaska.
12. N. Carolina.	12. Oregon.	12. Washington.
13. Georgia.	13. N. Carolina.	13. Other States.
14. Washington.	14. Alaska.	14. N. Carolina.
15. S. Carolina.	15. Georgia.	15. Georgia.
16. Other States.	16. S. Carolina.	16. S. Carolina.

Production of Gold and Silver in 1886.*

State or Territory.	Gold.	Silver.	Total.
	$	$	$
Alaska .	446,000	2,000	448,000
Arizona .	1,110,000	3,400,000	4,510,000
California	14,725,000	1,400,000	16,125,000
Colorado	4,450,000	16,000,000	20,450,000
Dakota .	2,700,000	425,000	3,125,000
Georgia .	152,500	1,000	153,500
Idaho .	1,800,000	3,600,000	5,400,000
Montana	4,425,000	12,400,000	16,825,000
Nevada .	3,090,000	5,000,000	8,090,000
New Mex.	400,000	2,300,000	2,700,000
N. Carol.	175,000	3,000	178,000
Oregon .	990,000	5,000	995,000
S. Carol.	37,500	500	38,000
Utah .	216,000	6,500,000	6,716,000
Washington .	147,000	80,000	227,000
Texas .	...	200,000	200,000
Other States	5,000	5,000	10,000
Total .	34,869,000	51,321,500	86,190,500

* Report of the Director of the U.S. Mint for 1887, p. 75.

The length of an hydraulic mining season depends for the most part on the water supply, which is always more or less limited by ice or drought. The large companies who own their own water and have such a supply of it that they can sell it to others, run all the year round, but some of the smaller ones who either have an insufficient supply or purchase water of other companies, and who do not have a very large surplus, can often be worked only from three to four months in the year. Most of the large mines work in the height of the season day and night full time with two shifts of ten hours each. The small mines usually work in the day only. When the water becomes scarce or when labour is difficult, all the mines are worked during the day only. The shift then is from ten to twelve hours. Many of the mines use large bonfires to illuminate their work during the night. The Spring Valley Mine used 8000 candle-power electric machines for the purpose of night working. It employs 240 men. The North Bloomfield Company also use the electric light, having put up a 12,000 candle-power machine for the purpose. The returns are, of course, greatest where the work is continuous, and in this, as in most every other case, the greatest capital commands the greatest amount of profit. The time when the work can best be done is, however, the very time when there is least water, and the season when there is the greatest supply is the least propitious for working. In the more important hydraulic mines throughout the country, the length of the season may be stated as about 200 days. This includes the average of those who work night and day throughout the year as well as those whose season is very short. During the latter part of the run most of the mines are worked on half-time or else with a diminished water supply. In some of the small mines the season is very much shorter even than this. Although the work in hydraulic mines is often laborious it is of a very healthful nature and is not more of a tax upon the men than the shorter shifts in underground mining. The wages are quite as high as in other occupations. The accidents likely to occur are less, as are also the risks. The liability to disease is less, the living is certainly as good, and altogether it seems to be a desirable rather than an undesirable occupation.

What the loss in gold is can only be estimated. The ordinary means of estimating what the loss is, fail here. The quantity contained in the original gravel is so small that even a very minute loss makes a large proportion of the total amount of gold contained. There does not seem to be any large quantity of it lost from being in the condition in which it could amalgamate, even if the sluices were extended indefinitely. Much has been written about rusty gold, or specks of gold covered with an outside coating of something which prevents the contact of the mercury, but that there is any considerable quantity of such material does not appear. It seems to be more the lightness of the gold, the rapidity of the stream and the impossibility of making contact between it and the mercury, either from too great rapidity of the current, which makes the small particles float, or from too much or too little water to carry off the gravel which is with it. The loss in gold is undoubtedly large, but it would often cost more to save it than it would be worth. By far the largest quantity of the gold lost is fine gold, some of it is undoubtedly rusty gold, or will not amalgamate, some of it is lost as small pieces attached to cement. Such pieces may be a double scource of loss, for should they attract quicksilver they may increase in value, and still not become heavy enough to be caught. A very small proportion only is in comparatively large pieces. There is no difficulty whatever in catching the large pieces, as they amalgamate easily; but a large part of the rusty and fine gold is lost. This cannot be much less than 15 to 20 per cent. of the total yield.

In most of the well-established hydraulic mines it is claimed that about 90 per cent. of the gold in the gravel is saved. This, however, is not based upon any scientific examination, but only upon a general estimate and a few assays. It is almost impossible to arrive at exact figures as to the amount of gold contained in the gravels, as it is unequally distributed, but it is quite likely that in the best mines this estimate is very near the truth. In some cases, generally where the number of sluices is not sufficient or the grade is too high, or there is some other defect in the working, or because the ground contains a large amount of clayey matter, the loss is probably as great as from 30 to 40 per cent.

It would appear, therefore, that the loss admitted is between 10 and 50 per cent. When the infinitely small amount of gold contained in the gravel as compared with the bulk of the material treated is taken into consideration, there seems to be very little doubt that the hydraulic mines, as a whole, work as close or even closer than the ordinary mill processes. In some investigations which I had occasion to make in the year 1876 on the working of the copper-dressing plants on Lake Superior,* I had occasion to show that a very considerable percentage of that metal was lost there, by the fact that the very thin flakes of it were attached to the rock, not enough to change in any considerable degree the specific gravity of the sterile material, which was then carried off with the other detritus. The same is true to a much greater extent with gold, because it is more valuable and smaller amounts of it are contained in the rock. It is all the more likely to be so because the water containing the gold is thick and muddy, and consequently of a higher gravity, and would buoy up small pieces of rock with the metal attached even better than at the copper mines. Where the quantity of material which rolls on the bottom is large, there is very little doubt that the amalgam after it has been once formed is often floured and carried off in that way. There does not seem to be any way of guarding against these difficulties except by the use of larger quantities of water. The effect of the rolling of the clay in the large sluices is exactly the same as in the small, and if the clay is not broken up with the drops or with the rush of water, this is undoubtedly the scource of a certain loss. That the loss is large, however, is shown by the fact that when the tails are public property, as they often are, they are sometimes re-worked with a profit by white men, but more especially by Chinamen. Whether they can be re-worked or not, there is always a considerable loss. This is owing to the general fact that too much gravel is washed for the supply of water. The difficulty of saving fine gold in large sluices in rapid streams is very great, even where the number and surface of the undercurrents is very largely increased over and above what is the normal working in California. Fine gold can only be caught by diminishing the

* Metallurgical Review, vol. ii., p. 227, 1878.

grade, and multiplying the number of undercurrents. It is expensive to do this, and no matter how many of them there are, the last will always catch something. Diminishing the grade limits the quantity that can be worked in a given time. It must be determined in every case what it will pay to save. At the present time it might not be economical, desirable, or even possible to separate all the gold, as it would undoubtedly cost more than it would be worth after it was saved. It will generally be found more profitable to treat a maximum quantity at a maximum loss, rather than a minimum quantity at a minimum loss. There is a large loss of gold in the first case, which, as a matter of political economy, it is shortsighted to lose, but if the loss was not made many of the mines would have to be closed.

The loss in quicksilver depends upon the rapidity of the current, the character of the gravel, the length of the sluices, and the time they are running. The longer the sluice the more carefully it is watched, and the greater the number of undercurrents, the less the loss. It is, however, very variable, and may vary from single to more than double, depending upon various causes. It has been known to fall below 10 per cent. and to reach as high as 37½ per cent. of the quantity used, and in by far the larger number of cases as high as 10 to 15 per cent. is known to escape. The loss of quicksilver involves the loss of gold, but in what ratio is not known. The amalgam from the undercurrents has been found to be poorer in gold than that from the main sluices.

The less carefully the sluices are watched and repaired, the longer and more rickety they are, the greater the loss. This loss is occasioned in part by the flouring of the quicksilver from the constant grinding of the large boulders which are carried along by the stream, the floured mercury being carried off with the water into the tailings, and partly by careless handling. This floured quicksilver sometimes collects in spots in the tailings, so that shining globules can, not unfrequently, be seen in the dump. They will occasionally run together in quite large quantities, and instances have been known where a number of pounds have been taken from a single pocket. As a general thing, however, it does

not collect, and now that the price of mercury has fallen to between $0.40 and $0.45, economy in its use does not seem to be a matter of so much importance as it was in 1874-5, when it ruled from $1.50 to $1.75 per pound.

AMOUNT OF QUICKSILVER LOST BY THE LA GRANGE AND NORTH BLOOMFIELD COMPANIES DURING THE YEAR 1874-75.*

Name of Claim.	Miners' Inches Used.	Length of Sluice.	Loss of Quick-silver.	Per Cent.
		ft.	lb.	
La Grange Co.†.	386,972	1800	900	11
Woodward Claim	51,550	600	217	25
Eisenbeck ,, ...	26,000	400	125	25

AMOUNT LOST DURING THE YEAR 1875-6.

Name of Claim.	Miners' Inches Used.	Length of Sluice.	Loss of Quick-silver.	Per Cent.
		ft.	lb.	
La Grange Co.	700,000	1800	2251	
Woodward Claim	30,000	600	123	
Eisenbeck ,, ...	56,200	400	182	

As the amount of quicksilver used is known both by its weight and cost, a careful investigation of the quantity of amalgam collected from different portions of the sluices and also that which escapes in the tails might give very interesting results. It would at the least indicate the ratio in which the gold that is susceptible to amalgamation is lost as compared with the waste of the quicksilver. As is well known by those who handle a great deal of quicksilver, it can be very easily floured and is then with very great difficulty restored to the liquid condition again. This is true, though to a somewhat less extent, with the amalgam, which has been known to have escaped in the shape of flour, and to have been collected in the stream twenty miles below the point where the sluices from which it had escaped had entered the stream.

The accidents which occur in hydraulic mines are owing to, first, caving of the bank, to which more than four-fifths of the

* Trans. Am. Inst. Min. Eng., vol. vi., p. 49.

† Each inch giving 2230 cubic feet per twenty-four hours.

accidents are due. Notwithstanding the fact that the bank always gives warning and that it is always possible to get out of the way, some of the miners are so venturesome that they are sometimes overwhelmed in the débris. Most of such accidents result fatally. The second class of accidents results mostly from the premature explosions of blasts, and are usually the result of carelessness on the part of the men. The third are those which result from the attempt to undermine and roll forward boulders, rocks, stumps, and logs, which are in the way. A fourth cause is the careless handling of the tools and machines. Taken altogether, all the accidents through the year are exceedingly small in comparison to the number of men employed, and shows both great skill and judgment on the part of the men. There are few methods of mining where the proportion of fatal accidents is so small.

There are many ways of estimating the value of deep placer ground in California.* These are, first, the average yield of a cubic yard of ground worked during a long time; second, the yield per miners' inch over a considerable period; third, the yield per superficial yard or acre of bedrock stripped. As the height of the bank is not uniform this method is of local application only, and applies to districts where the bank is of nearly uniform height. Fourth, in the drift mines the estimation is made per yield of carload, per cubic foot, or per lineal foot of channel. The system of valuation per cubic yard does not offer any opportunity for comparison between the different claims on account of the disparity of conditions and the variable nature of the material treated. In all very deep placers the difference between the top and lower gravel may be very great, so that a few feet can only give an approximate idea of the value of the claim. The estimate per miners' inch is a much more accurate method of stating the value. Custom has decided that the calculation of the product and cost of hydraulic mines shall be by the cubic yard of gravel washed, and this is likely to remain the standard in spite of all its deficiencies. For technical purposes, as we have seen, the product of a miners' inch of water used is a familiar means of comparison, and is preferred by most engineers, and would, if it

* Prod. Gold and Silver in the U.S., 1880, p. 318.

could be adopted, be generally preferred. From a business point of view the expenses and profits are best shown by reference to the quantity of gold produced at its market value. The cost of producing a dollar in gold may be stated in the value of labour, water, powder, quicksilver, or, in other words, in items of what has been actually expended to get it, each item forming a percentage of the product, and the advantage of this method is that it admits of comparisons, with results obtained in other branches of mining, as well as between the hydraulic mines themselves.

The profits, however, do not depend so much on the yield per unit adopted as a method of estimating the value, as on the skill and economy with which the great mass of material is worked, the cost of water, the scale on which washing is adopted, and good business management. It is doubtful if the North Bloomfield Mining Company could have extracted the gold at a cost of only two cents to the cubic yard if they had not had all these conditions fulfilled and owned their water besides. The examples cited below are from the records of companies who have treated low-grade gravels. Their average yield per cubic yard was 7½ cents. With a bank 45 ft. high the yield would be $5263.50 per acre. It was found after a careful examination of the upper ground in the district of Gold Run, Placer County, that the yield per cubic yard was about 0.0475.* At French Corral, Nevada County, the average yield of the upper ground was 15 cents; of the cement and lower strata $3.50. The cost of working the cement was $1.10, yielding a profit of $2.40. The average yield of the North Bloomfield Mine has been from the top dirt down about 10 cents per miners' inch of twenty-four hours. It has yielded as high as from 13½ to 27½ cents. The average of six years from 1874 to 1879, of the Paragon Claim, Placer County, has been per miners' inch of twenty-four hours, with 6-in. pressure, $2.54 for the whole height. The highest yield was $4.04 and the lowest 86 cents. The amount of gravel moved varies with the hardness of the ground and the force of the water. In large mines, such as North Bloomfield, Spring Valley, Excelsior, and others, the head or pressure of water was 150 ft. to 300 ft., the diameter of the nozzle from 6 in. to 8 in. Such a stream will carry from 600 to

* Min. Com. Rept., 1872, p. 119-120.

800 in.* of water, and each twenty-four hour inch on an ordinary grade will work from five to seven cubic yards of gravel, weighing about $1\frac{8}{10}$ tons per cubic yard. At North Bloomfield they calculated that it took 19 cubic feet of water to move one cubic foot of gravel in the bank.

VI. The Treatment of the Tails.

Most of the experiments made upon the working of the tailings of hydraulic claims have resulted unsuccessfully in a commercial point of view. At Gardner's Point, for instance, all the tailings were caught and re-worked to ascertain what the loss was, and resulted in catching not over 1 per cent. of the yield of the first washing. There are cases, however, where the tailings have been successfully worked. At Slate Creek, in Sierra County, California, on one of the branches of the Yuba River, a deposit of tailings had accumulated for more than thirty years. The deposit is six or seven miles long with a depth ranging from 1 ft. to 35 ft. It varies from 100 ft. to 800 ft. in width, but has an average of about 200 ft. It has a uniform grade of $15\frac{1}{2}$ ft. to the thousand. These tails are estimated to be worth 10 cents to the cubic yard. They are the result of the washing of hundreds of acres of ground. These deposits are, for the most part, the tailings from sluice working of very rich shallow placers, and of some veins, and contained a considerable quantity of mercury. There is here free water sufficient, and several companies have worked successfully upon them. This must, however, be considered as an exception.

It has been proposed to work such tails by making an open cut in the bedrock at the lower end of the deposit and running it up year by year until the grade should run out. The tails are thrown into the cut by wing dams erected at intervals. The rock cutting is a ground sluice and is terminated by ordinary paved flumes and undercurrents. This system is in successful operation in Indian Cañon, Placer County, below Iowa Hill, and others have been projected on Bear and other rivers.†

When the tails from the other mines are re-worked, the

* Prod. Gold and Silver in the U.S., 1880, p. 320.

† *Ibid.*, 1884, p. 535.

apparatus for doing so are ground sluices, flumes, and undercurrents. After these are once set up the expense of working is extremely small. They are simply for supplies and repairs for the sluices and for watching them to prevent others from cleaning them up. They are not usually cleaned oftener than once in the season and are then got ready for the next season's work. Below Gosling Ravine a flume 4200 ft. long, 16 ft. wide, and 7 ft. high, divided in the centre by a partition and terminated by a double undercurrent, was constructed with shorter sets of flumes and undercurrents below. The profit realised by the working was $30,000 in 1871, $34,000 in 1872, $24,500 in 1873, and $25,290 in 1874, or a total of $113,790. The entire cost of working during this time was $18,000, leaving a net profit of $95,790 for four years' work. There is no doubt that all these accumulations of tails contain besides the float gold considerable quantities of quicksilver and amalgam, and that their working would in many cases be very profitable if a proper outlet and economical method of working them could be found. The ordinary yield in the northern counties has been found to be from $40,000 to $50,000 per acre. In an exceptional case at French Corral, in Nevada County, it reached the enormous sum of $200,000 per acre. In the southern counties, where the gravels are not so deep, it has varied from $10,000 to $40,000 per acre. Taking the basis of a miners' inch, the yield has sometimes reached as high as a dollar per inch.

The tailings, though they enter the public domain, are mostly too poor to be worked. It may be that with the concentration which nature effects, they may some time be worth working. There is no doubt but that in the future new methods will be devised for more economic treatment, just as the necessity of the case caused the invention and perfection of hydraulic mining which was, up to that time, unknown. In the mean time, with all its defects, commercially speaking, hydraulic mining must be regarded as the perfection of working deep placers and low-grade gravels on a large scale.

It has been found by experiment that a cubic yard of ordinary gravel to be hydraulicked will vary in weight from $1\frac{1}{2}$ to $1\frac{3}{4}$ tons. This material is handled in mines like the North Bloomfield Mine at from 2 to 3 cents per cubic yard, which seems wonderful

until the efficiency and economy of hydraulic mining is studied. If a ton of gravel is treated at a cost of 2 cents, the expense is between one-fiftieth and one-hundredth of the minimum outlay involved in mining and milling the most favourably situated and most easily reduced free milling gold ores. The bulk to be treated, however, is immeasurably larger. It may be stated that the average cost of hydraulic mining bears about the same relationship to the average cost of treating gold ores by mining and reduction as does this minimum. In making a comparison between hydraulic mining and the reduction of rebellious silver ores, the difference is still greater. If the two extremes are stated as 2 cents for the hydraulic gravel on the one hand, and $100 per ton for mining, hauling, and reducing silver ore under exceptional disadvantage, the range of cost of handling similar weights of material is expressed by the ratio of 1 to 500.* In addition to the previous advantages possessed by the hydraulic process, such as saving in labour, &c., and the capability of handling large quantities of material with great rapidity, a feature which especially characterises it, is the fact that it is continuous and very nearly automatic throughout, while in vein mining the process is divided into the distinct operations of mining and reduction, with possibly many successive handlings in the different stages of treatment. The minimum cost of hydraulic mining taken from the accounts of a few of the largest and most favourably situated and most economically and skilfully worked mines is apt to be misleading, if applied as a basis of calculation in projecting new enterprises. While a few exceptional mines, like the North Bloomfield, are worked at a cost of between 2 and 3 cents per cubic yard, it is probable that the average cost of all the hydraulic mines of the country, large and small, is not less than 10 cents per cubic yard, and there are very exceptional cases where it is as high as 40 or 50 cents per cubic yard. As is seen by what is stated above, the economy of this method of mining depends for its success on handling enormous quantities of gravel of a very low grade. In many mines the maximum product is not over 10 cents per cubic yard, and in some favourably situated workings a considerably lower rate

* Report of U.S. Census, 1880, vol. xiii., p. 201.

results in a profit. Between 1870 and 1874 the North Bloomfield Company washed 3¼ million cubic yards of top gravel, yielding 2.9 cents per cubic yard, at a profit of $2232.84. If there had been a very slight improvement in the quality of the gravel there would have been a considerable profit. In 1880 they were able to pay the cost of working of 2½ cents per cubic yard without a profit. The conditions for cheap working are: first, ground that will pay; second, an abundant supply of good water; third, sufficient dump; and fourth, gravel sufficiently loose to be piped without the expenditure of large amounts for blasting. An essential requisite in these is good business management and the strictest economy in every detail of the work.

The following Table gives a comparative statement of expenses and yield of the No. 8 Mine of the North Bloomfield Gravel Mining Company for the season of 1874-5 :*

Runs.	Water used.	Time occupied.*	Yield per Inch.	Gross Yield.	Expenses.	Profit.
	in.	days.	$	$	$	$
First .	61,000	64	.14¼	8,700.00	8,475.00	225.00
Second .	58,875	46¾	.159	9,360.00	7,850.00	1,510.00
Third† .	47,538	29¼	.217	10,300.00	6,150.00	4,150.00
Fourth .	54,198	26	.155	8,550.00	5,600.00	2,950.00
Fifth‡ .	59,652	61	.204	12,150.00	11,150.00	1,000.00
Sixth .	52,134	33	.234	12,200.00	6,900.00	5,300.00
Seventh§ .	53,575	35	.244	13,011.77	6,963.83	6,047.94
Total .	386,972‖	295¶	.193**	74,271.77	53,088.83	21,182.94

* "Time occupied" includes every day from the time washing commenced until washing commenced on next run.

† This run washing was chiefly up Hilder Ravine, where bank had very little of top or poorest gravel.

‡ During this run the timbers and blocks were washed out of shaft 8, causing a delay of over three weeks, and an extra expense of, say, $3000.

§ The clean-up for this run was $15,038.06 ; expenses, $6963.83 ; showing a profit of $8074.23. As, however, this clean-up was a final one, all the sluices being cleaned, a portion of it properly belongs to the preceding runs, and has been credited to them in above statement for the purpose of making an exact comparison.

‖ Inches for twenty-four hours.

¶ From December 21, 1874, to October 11, 1875.

** Average.

* Min. Com. Rept. for 1876, p. 93.

MINING AND WATER COST AND YIELD.

——	Labour.	Supplies and Tools.	General Expense.	Total Costs, Water and Mining.	Bullion Products.	Yield per Inch of Water.	Net Profit.
	$	$	$	$	$	$	$
No. 8 Mine	22,790.39	12,489.09	3,328.95	53,088.83	74,271.77	.192	21,182.94
Woodward ,,	1,982.00	625.00	250.00	4,786.00	6,011.60	.117	1,225.60
Eisenbeck ,,	1,459.75	550.00	100.00	3,086.35	2,725.06	.104	361.29
Hilder Tunnel	...	...	...	45.00	70.20	...	25.20
Total .	26,232.14	13,664.09	3,678.95	61,006.18	83,078.63	...	22,072.45

The following Table gives some of the details of the hydraulic claims, Calaveritas District, Calaveras County, Cal. :*

——	Bennet's Claim.	Johnston's Claim.	Hedrick's Claim.
Inches of water used in hydraulic washing	40	40	40
Cost of water	$20 per week.	$20 per week.	$20 per week.
Fall of the water (head) . .	30 ft.	80 ft.	80 ft.
Supply of water lasting in a year .	4 months.	4 months.	3 months.
Number of cubic yards washed in ten hours	3 to 4	3 to 4	3 to 4
Yield of gold per day . . .	$10 to $12	$12 to $14	$15
Height of gravel washed this year	12 ft. to 14 ft.	10 ft. to 15 ft.	20 ft.
Ground worked in the claim, total	1 acre.	5 acres.	2 acres.
,, unworked. . . .	5 acres.	15 acres.	60 acres.
Ground worked this year . .	40 ft. by 50 ft.	70 ft. by 70 ft.	40 ft. by 100 ft.
Number of men working . .	5	3	5 to 7
Wages paid to hired men per day, per man	$1.25	$1.25	$1.25
Composition of auriferous deposit, from top	Red soil, round heavy quartz boulders of all colours, lava sand, silicious slate, fragments, white rounded quartz, granite, syenite, boulders, gneiss, fine gold, iron sand and rubies.	Red soil, round heavy quartz boulders of all colours, lava, sand, fragments of slate, gneiss, granite, syenite, fine gold, magnetic iron sand.	Red gravelly soil, sand, quartz boulders, cemented fragments of slate and other rocks, gold, fine black sand, &c.
Direction of channel . . .	N.W., S.E.	N.W., S.E.	N.W., S.E.
Bedrock	Blue and grey slate and strata of quartz.	Blue and grey slate.	Blue and grey slate.
Curiosities found	...	Petrified turtles.	Petrified wood and foliage.

* Min. Com. Rept., 1872, p. 72.

The following Table gives the itemised cost of mining during the year 1880 of a number of the principal hydraulic mines in the northern part of California. They are the results of the operations of a group of mines in Stanislaus County and at the North Bloomfield Mine, Nevada County.

COST OF HYDRAULIC MINING AT VARIOUS MINES IN DEL NORTE AND SISKIYOU COUNTIES, CALIFORNIA.*

Cost of—	Bunker Hill.	China Creek.	Happy Camp.	Muc-a-muc.	Wingate.	Bay City.	Coyote Gulch.	Thompson's Creek.
	$	$	$	$	$	$	$	$
Labour . .	4950.00	2167.36	8,588.00	1039.00	4288.50	835.00	1950.00	1095.00
Repairs to ditch .	..	...	1,000.00	56.00				
Powder and fuze .	2250.00	315.00	500.00	135.00	800.00			
Blocks, &c. .	1000.00	412.80	1,587.00	110.00	500.00	100.00	24.00	40.00
Boxes . .	60.00	120.00	...	...	60.00	100.00	30.00	
Quicksilver. .	150.00	45.00	25.00	12.00	35.00	...	14.00	28.00
Tools . . .	50.00	100.00	100.00	...	50.00	...	50.00	
Sundries . .	200.00	20.00	...	148.00	200.00	...	...	100.00
Total . .	8660.00	3180.16	11,800.00	1500.00	5933.50	1035.00	2068.00	1263.00
Number of cubic yards of gravel moved . .	132,300	32,234	314,600	22,554	263,000	24,750	31,740	19,600
Cost of moving 1 cubic yard .	0.0654	0.0987	0.0375	0.0665	0.0225	0.042	0.065	0.645

The average cost per ounce of gold and silver produced during two years by the Lagrange Company is given:

Cost of a Troy Ounce of Metal.†

	cents.
Water	1.43
Labour	6.85
Material	1.81
Officers	0.94
Contingent expenses	0.26
Taxes	0.09
Total cost of metal produced ...	11.38

* Report of U.S. Census, 1880, vol. xiii., p. 202.

† Trans. Am. Inst. Min. Eng., vol. vi., p. 99.

Cost per Cubic Yard Treated.

	cents.
Water	0.008
Labour	0.036
Material	0.010
Officers and contingent expenses	0.006
Total cost of metal produced ...	0.060

The average yield of the cubic yard was $0.1019. The average amount of gravel moved per miners' inch was 1.48 cubic yards; the value of the ounce of metal in gold and silver was $19.29. These items of expense are given for light pressure and small grades.

The following Table gives the statistics for an irregular period up to the year 1876:

COST OF HYDRAULIC MINING AT VARIOUS MINES IN STANISLAUS AND NEVADA COUNTIES, CALIFORNIA.*

Cost of—	Chesnau.	French Hill.	Johnson.	Light.	North Bloomfield.	Sicard.
	$	$	$	$	$	$
Labour. . .	0.036	0 042	0.02	0.022		0.025
Blocks and lumber	0.0042	0.0042	0.0029	0.008		0.0061
Material . .	0.0062	0.0057	0.0076	0.0015	0.0225	0.0038
Melting and refining . .	0.0065	0.0004	0.0004	0.0003		0.0007
Water . . .	0.008	0.0104	0.006	0.006	0.0075	0.004
Total cost per cubic yard .	0.055	0.063	0.037	0.038	0.03	0.039
Yield per cubic yard . . .	0.16	0.13	0.04	0.066	0.0533	0.13
Yield per inch of water . .	0.23	0.14	0.08	0.12	0.23	0.37
Number of cubic yards of gravel washed . .	284,932	676,968	196,632	683,244	4,104,700	155,347
Total yield .	47,781.73	90,186.19	9,148.27	45,444.65	267,007.50	20,197.07
,, cost .	15,923.71	42,655.83	7,465.99	25,962.82	147,912.58	6,205.40

For heavy pressure and 4 per cent. grade the average cost of the gold at the North Bloomfield Company's mine during 1875-6 was:

* Report of U.S. Census, 1880, vol. xiii., p. 202.

*Cost of One Ounce of Metal.**

	$
Labour	3.93
Blocks and lumber	0.50
Explosive	0.98
Materials	0.88
General expenses	0.70
Water	2.09
Total...	$9.08

The value of the metal in gold and silver was $18.53.

The total expenses at North Bloomfield for 1875-6 were:

	$
Labour	40,975.85
Water...	21,740.97
Explosives	10,279.73
Material and supplies	9,249.96
General expenses	7,364.12
Lumber	5,212.62
Total	$94,823,25

The total expenses of this company for 1871-2 were:

	$
Labour	20,500
Water	9,700
Materials used in construction	8,600
General expenses	6,200
Total expenses	45,000
,, amount of gold produced	54,000
Profit	$9,000

If during this year the company had simply sold their water at the rate of 16 cents an inch for twenty-four hours, they would have had a profit of $40,000. From the year 1866 to 1879, the total yield of this company was $1,435,680.47, of which $595,444.36 was profit.

The expense and yield of various mines previous to 1874 were as follows:†

* *Ibid.*, p. 100.

† Annales des Mines, 7 Series, vol. ix., p. 65.

—	Cost per Cubic Yard.	Yield per Cubic Yard.
	cents.	cents.
Roach Hill	6	60
Richardson	3	15
Iowa Hill	2.5	71
Independence	2	25
Wisconsin	2	12.5

After the first run of the mine in Smartsville the following results were obtained:

Time of running the water...	40 days
Amount of gravel washed daily*	1500 cubic yards
,, water used daily	750 miners' inches
Quantity of earth moved per miners' inch and per day of ten hours	2 cubic yards
Quantity of earth moved per hour	½ cubic yard
,, gold collected daily†	$665
Total quantity of gold	26,000
Quantity of gold per miners' inch per day ...	0.88
,, per cubic yard of dirt	0.44

700,000 miners' inches were used, and 2,919,700 cubic yards of gravel moved, making 4.17 cubic yards per inch of water. The total yield was $192,735.73.

The following Tables give the statistics of claims in Stanislaus County, California, working low-grade gravels during two seasons:‡

Name of Claim.	Average Height of Bank.	Pressure of Water.	Cubic Yards Washed.	Cubic Yards of Gravel moved per Miners' Inch.	Yield per Miners' Inch, Water.	Yield per Cubic Yard.
	ft.	ft.			cents.	cents.
French Hill Claim .	30	50 to 70	676,968	1.08	14	13
Light Claim . .	45	60	683,244	1.80	12	6⅔
Chesnan Claim .	28	70	284,932	1.37	23	16
Johnson . .	30	80	196,632	1.76	8	4
Licard . . .	90	90	155,347	2.89	37	13
For a corresponding period the North Bloomfield, of Nevada County, showed the following results	180 to 260	...	4,104,700	4.5	23	5

* A flow of one cubic foot per second will give 38 miners' inches.

† The quantity of gold in 1874 was $800 daily.

‡ Prod. Gold and Silver in the U.S., 1880, p. 319.

Name of Claim.	Cost of Water per Cubic Yard.	Cost of Labour, Material, &c., per Cubic Yard.	Total Cost per Cubic Yard.	Total Expenses.	Total Yield.
	cents.	cents.	cents.	$	$
French Hill Claim .	1.	5	6	42,655.83	90,186.19
Light Claim. . .	.006	3	3⅜	25,962.82	45,444.65
Chesnan Claim . .	.008	4	4⅜	15,923.71	47,781.73
Johnson . . .	.006	3	3⅜	7,466.00	9,148.27
Licard	.004	3½	3 9/16	6,205.40	20,197.07
For a corresponding period the North Bloomfield, of Nevada County, showed the following results	.0755	2¼	3	147,912.58	267,007.50

The following Tables give the statistics of the hydraulic mines of the Forest Hill Divide, Placer County, California, for 1879:*

Mines.	Inches of Water.	Pressure in Feet.	Yards moved per Inch.	Yield per Yard in Cents.	Yield per Inch in Cents.
Hobson . .	42,789	360	20.6	3.50	70.
Pritchard . .	51,015	{100 / 200}	2.73	10.71	26.48
Morning Star .	139,759	450	5.61	8.87	37.24
Orion . . .	42,000	275	10.19	2.48	50.00
Enterprise . .	46,362	325	5.61	8.78	49.24

TABLE SHOWING THE YIELD OF GRAVEL PER CUBIC YARD OF HYDRAULIC MINES IN CALIFORNIA.†

Name of Claim.	Location.	Yield per Cubic Yard.	Authority.
		$	
American Company.	Sebastopol, Nev. Co. .	0.24	Hamilton Smith, Jr.
Blue Tent . .	Nevada County. .	.15	R. W. Raymond.
Blue Point, 1870 .	Smartsville, Yuba Co.	1.23	Hamilton Smith, Jr.
Gold Run, 1870 .	Placer County . .	04.8	William H. Pette.
Dardanelles and Oro	Forest Hill, Placer Co.	13.1	Joseph McGillivray.
McCarty's Diggings.	Colum. Hill, Nev. Co.	43.3	J. D. Hague.
Smartsville Mine .	Sucker Flat, Yuba Co.	19.5	Amos Boman.
Union Gravel Mine.	Empire Hill . .	15	,, ,,
Pactolus ,, ,, .	Yuba County . .	20.8	,, ,,
Blue ,, ,, .	,, ,, . .	63	,, ,,
Pittsburg . .	,, ,, . .	41	,, ,,
Pactolus. . .	,, ,, . .	44	W. Ashburner & J. D. Hague.
North Bloomfield .	,, ,, . .	12.0 to 16.3	Hamilton Smith, Jr.
Spring Valley. .	Cherokee, Butte Co.	28	R. H. Stretch.
French Corral. .	Nevada County. .	33.4 ,, 47.4	Hamilton Smith, Jr.
Manzanita . .	,, ,, . .	27.4 ,, 33.3	,, ,, ,,

* *Ibid.*, p. 321. † *Ibid.*

The enormous extent to which this method of mining has been carried on is shown by the Table below,* which gives the area in acres of some of the large claims.

	County.	Acres.
Spring Valley	Butte	1215
Morris' Ravine...	Butte	1300
Happy Camp	Del Norte	507
Liberty Hill	Nevada	554
North Bloomfield	Nevada	1585
Gold Run	Placer	500
North Fork	Plumas	800
Plumas...	Plumas	1355
Buckeye	Trinity	1700
Lewiston	Trinity	640
Weaver's Creek	Trinity	600
Golden Rock...	Tuolumne	640

The ground of these mines in Colorado, Idaho, and Dakota is held in large blocks by companies. The Castle Creek Mine in Dakotah has eighty-eight claims 300 ft. in length, with the entire width of the deposit whatever that happens to be. The title of the miners was that of discovery, and consists of what is known as "miner's title." As these were for very small claims they were afterwards consolidated. They then secured patents from the United States. Generally the original locators did not procure patents. It appears from the United States Census that most of the ownership is in the State or Territory where the mines are located, and that only 12 per cent. are controlled by foreign capitalists, and of this nearly one-half is held in the State of New York.

Investment and Ownership.

The nominal capital stock† of the mining corporations has but little significance. The placer properties are usually capitalised at much less amounts than has been the custom with deep mines though for no apparent reason, unless, indeed, the fact that the stock of the former is less apt to be found upon the lists of the mining boards and is not so frequently used as a vehicle for

* Report of U.S. Census, 1880, vol. xiii., p. 186.
† *Ibid.*, p. 184.

speculation. But even thus the nominal capital has often little relation to the amount of money invested or the actual value of the properties. The present Spring Valley Hydraulic Gold Mining Company, for instance, has a nominal capital stock of only $200,000, in 200,000 shares of $1 each, though when placed upon the market the shares brought $10 each. In this case the incorporation was effected under the laws of the State of New York, the purpose of the low capitalisation being to insure individual stockholders against any possible personal liability or assessments. Thirty-six placer mining companies reported have a nominal capital of $35,115,000, or an average of $975,417, and the average par value of the shares of $34,815,000 of this amount is $14.68. There are a large number of claims held by unincorporated companies, each consisting of a few individuals. The ownership of these is often divided into a small number of shares, commonly the same number as that of the original holders, but each share of correspondingly higher value.

The amount of capital actually invested in placer mining cannot be definitely ascertained or even approximately estimated. In the case of the State of California *versus* the Gold Run Mining Company, the statement was made that the capital invested in mining in California is about $160,000,000, of which $100,000,000 is in gravel mining. Referring to this testimony Lieutenant-Colonel Mendell* remarks:

This appears to be a generalised opinion, based upon a partial knowledge of investments. It is not based upon detailed information, and is not claimed to be more than an approximation So far as it applies to gravel mining, the amount is probably in excess. Perhaps it is quite as well for the purposes of this report to be able to say that invested capital is large as to be able to give the specific amount. This kind of information for a majority of mining properties is quite out of the question.

For the following named companies Lieutenant-Colonel Mendell† gives the several amounts, including in each case presumably the cost of ditches, reservoirs, &c.

* Mining Débris in California Rivers, p. 17.

† *Ibid.*, p. 18.

Capital Invested in Leading Hydraulic Mines.

Name of Company.	Amount.
	$
Excelsior, of Smartsville...	2,857,166.55
North Bloomfield and Milton	4,079,321.62
Gold Run	433,335.00
Blue Tent	755,000.00
El Dorado Deep Gravel	1,008,611.60
Eureka Lake (approximate)	2,500,000.00
South Yuba	2,000,000.00
Spring Valley	2,000,000.00
La Grange (about)	650,000.00
Total	$16,283,434.77

When a mine is fully developed, the returns are very regular and uniform* and can be counted on with quite as much certainty as, for instance, in manufacturing pursuits. Indeed, it is usually possible to estimate beforehand with considerable precision whether the returns from a given undertaking are likely to pay for the expense of opening the mine. The chief elements of uncertainty are the amount of rainfall and the duration of the working season, which in all localities are liable to vary from year to year. These remarks, of course, apply mainly to claims having deep beds of gravel and worked upon a somewhat large scale, and in such mines, while there is not the constant excitement dependent upon the possibilities of sudden strikes and bonanzas, there is, on the other hand, little fear of the mine "petering" so long as there is a good gravel bank in sight and the means of working it are at hand. Hence it is that few hydraulic mines are worked at a loss, except during the prosecution of dead-work necessary to development, or perhaps, in unexpectedly short seasons. The hydraulic miner can easily ascertain whether it is worth while to continue operations, and when he finds himself running behind he is apt to close down at once. The amount of work done and of money invested in this class of enterprises is therefore very closely adjusted to the possibilities of profit. While, however, there are a large number of mines operated on a bare margin of profit, it is believed that, leaving out of consideration the value of the permanent investment, the running expenses average hardly more than 50 per cent. of the gross yield.

* Report of U.S. Census, 1880, vol. xiii., p. 202.

From the inception of hydraulic mining the tendency has been constantly toward improvement in method and increase in the scale of operations, necessitating the employment of large capital, and consequently, a consolidation of interests. At present the more important enterprises are prosecuted by strong corporations, which are able to perform an immense amount of preliminary dead-work before receiving any returns; but, on the other hand, many small mines are still yielding handsome profits on smaller investments to individual owners. Progress has been made chiefly in an extension in the field of operations, larger tracts being developed and heavier rigs brought to bear. Greater amounts of water, at higher heads, and nozzles of larger aperture; long tunnels, extended strings of sluices, and the use of double lines of sluices; increase in the number of undercurrents, drops, grizzlies, &c.—these are the principal directions in which advance is continually manifesting itself. Each year brings new knowledge, and though the hydraulic method has again and again been declared to be as near perfection in point of practical economy as could be hoped for, still there is reason to believe that, while the general principles are now well established, the future has in store many further improvements.

Hydraulic Elevators.

The shallow placers and modern river beds furnished nearly all the gold produced in California from its first discovery in 1848 up to about 1852 or 1853. After the discovery of the ancient river channels and the development of hydraulic mining, the treatment of shallow placers was almost entirely abandoned, not because they were considered to be exhausted, but on account of the expense of bringing water to them and the difficulty of finding a sufficient grade to get rid of the tails and deposit them where they would not be an incumbrance. In many instances the deposits to be treated are below the level of the ancient river channels, which have been broken by the streams of the present period, and the gravel swept by the great torrents of water to some depression on the line of the flow, where in lodging it formed what are now called "flats."* In this way, the erosion

* Prod. Gold and Silver in the U.S., 1880, p. 226.

of Table Mountain channel of Tuolumne County has created an immense deposit of auriferous gravel on both sides of the basalt-capped river, which once drained Central California. The depth of these deposits varies from 20 ft. to 80 ft., but owing to the want of grade long tunnels were always required to bottom them and to drain the water, and many thousand acres of this class remain unworked, awaiting the consolidation of claims and the necessary capital to run the tunnels. Great areas are found on the foothill benches of the Sierra Nevadas, and sometimes on the edges of the San Joaquin and Sacramento Valleys, at an elevation of from 200 ft. to 300 ft. above the level of the sea. At Bangor and Wyandotte, higher up on the foothills, they also occur. Below Oroville, Butte County, near the banks of the Feather River, there are many hundred acres of this shallow deposit. Much of this ground is in many cases worked by Chinese, who have been known to pay as high as $5000 an acre for the fee simple. In the Sacramento Valley near Pino on the line of the Central Pacific Railway, many hundred acres of this kind of auriferous gravel 3 ft. to 5 ft. in depth are found on the flat lands. In many cases these placers are deposited below or only slightly above the level of the river, and in winter, or when the river is high, they are entirely covered with water. In a few cases the bluffs near these deposits contain gold, so that it is very desirable to work out the flat lands before touching the bluffs, in order that the bluff tails may be deposited on the worked-out ground of the flat. In some few cases the level was sufficient to work off the top pay dirt, but the richer part below was left because there was no method then known of working it, and it is now covered with the accumulated débris from the sluicing of the gulches above, which have tailed over it and left the abandoned rich gravel covered deeper than before. How to overcome the difficulty of washing the pay dirt which either had no grade to carry off the tails, or else, like these placers, are below the level of the ordinary drainage of the country and consequently liable at any moment to be flooded with water, has been a matter which has occupied the attention of many engineers. A large number of experiments have been made both in this country and Australia to find the best method of

treating them. Such questions in the early days were, however, not discussed. Ground of this kind was considered to be unworkable unless it contained pay dirt rich enough to yield largely by the laborious treatment of the pan, the cradle, or the tom. The water for their treatment, as well as the seapage water, was

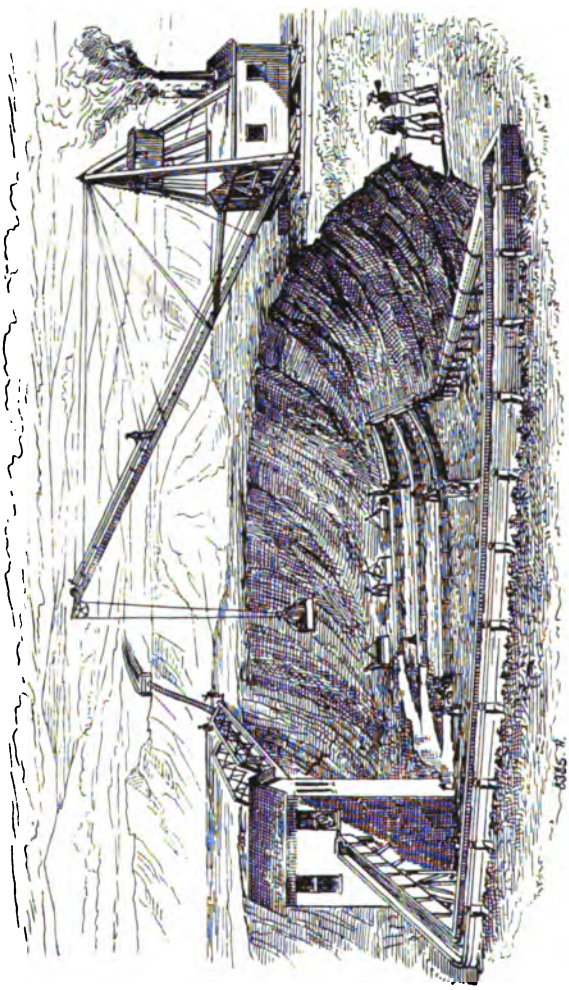


FIG. 35. MINING BELOW WATER LEVEL AT ARROYO SECO.

raised to the surface by means of Chinese pumps worked either by hand or flutter wheels. As the gravel was worked by hand the output was always small. When the surface ground had been worked but a small depth, it became necessary to abandon the claim before the richest deposits had been reached. It has

become possible within a few years to bedrock such deposits, and such lands are now successfully worked since the invention of dredges, sand pumps, and the hydraulic elevator, which are some of the marvellous but very simple things which characterise the inventive genius produced by the extraordinary conditions which also created hydraulic mining. The simplest of these is the method employed at Arroyo Seco Mine, at Dry Creek, three miles from Ione, Amador County, California, Figs. 35 and 36. The stream is running on its modern bed, but its ancient channel is rich in gold. The pay streak on the bedrock is from 2 ft. to 10 ft. deep. The bedrock is slate, which the gold frequently penetrates to the depth of three to four inches. The rich is covered by a variable amount of poor dirt, which must be removed before the pay can be reached, which at the top usually yields only a few cents to the pan, but on the bedrock often as many dollars. There is no fall, and, consequently, no way of disposing of the tails. Water can be had from a sufficient elevation to wash down the banks, but there is not much of it, and any plan involving its application must provide for its very economical use. To effect the stripping the poor dirt was formerly shovelled on to a wooden platform capable of carrying 5 to 10 tons of it. This was hoisted 50 ft. on a derrick, swung around and dumped, and the platform returned in about a minute and a half. Besides the men filling, this arrangement of the derrick required an engineer and one man to control the movements of the boom, and one man to dump.

The work of stripping is now done by steam crabs attached to two derricks capable of moving a load of 10 tons 50 ft. high. These are shown in Fig. 36, where one of these crabs is shown open as if about to descend to take a load, and the other closed and lifting one. This method of dredging was invented by Mr. J. P. Lambling, the present superintendent of the mine. The machinery for moving the crabs is all placed in a boom derrick car on wheels, which, when necessary, can be readily moved from place to place. The car is 56 ft. long and 16 ft. wide and 18 ft. high. It is covered on the outside with corrugated iron. It is prevented from turning over when the boom is at right angles to the car by two beams which project at right angles to

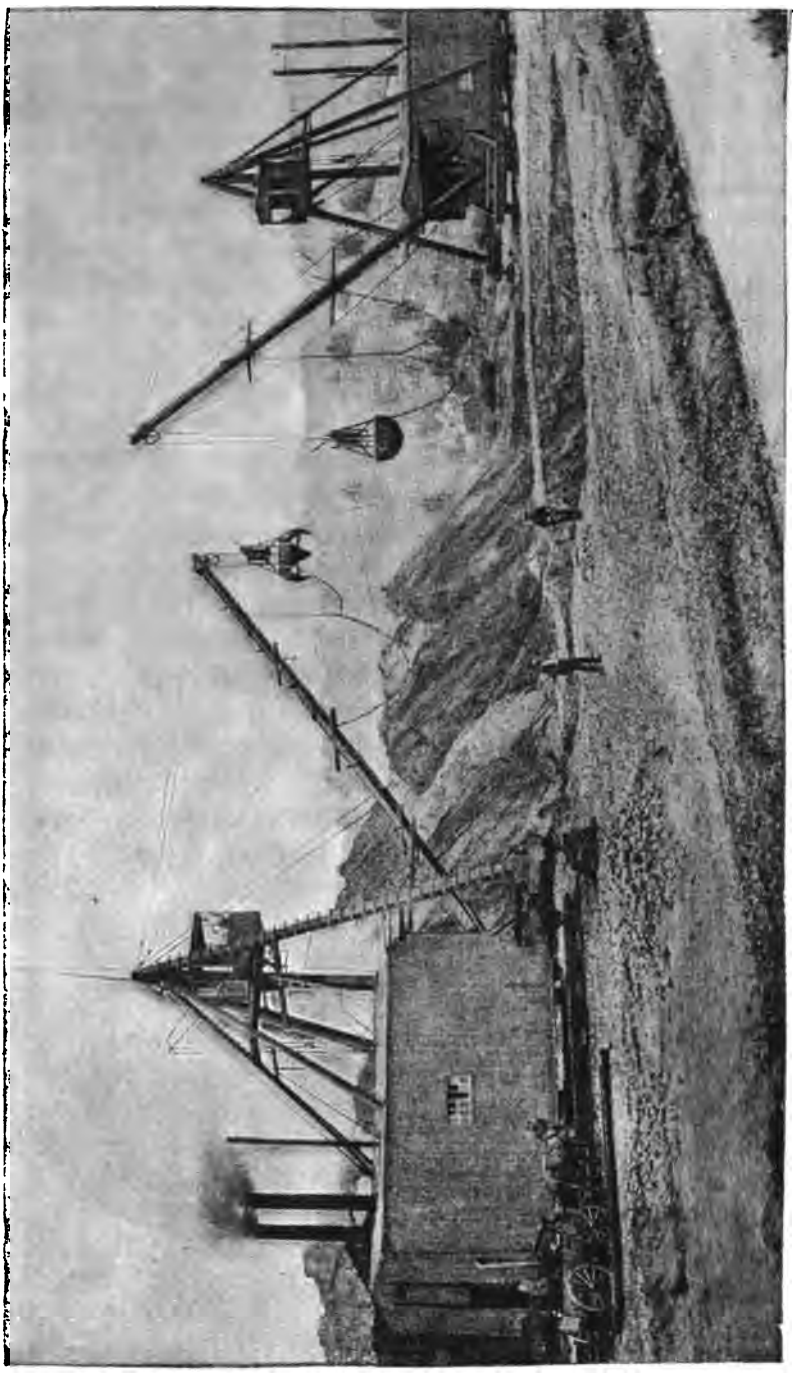


FIG. 36. DREDGERS AT THE ARROYO SECO MINE.

the car, thus giving it more than twice its base. The legs of these beams are long hand screws, which can be adjusted to any position. When the working of the pit makes it necessary to alter the position of the car, it is moved on the track, which can be taken up in front or behind so as easily to change it. On the front end of the car is a mast 56 ft. high. To this a boom 110 ft. long is attached, which can be moved sidewise and up and down at will. In each car there are two engines of 30 horse-power which use 4½ tons of inferior lignite in twenty-four hours. The total weight of the car is 75 tons. A self-feeding bucket or crab with steel jaws is hung to a steel wire rope, which is attached to the drum of the winding engine. On the top of the dredge or crab is a steam or compressed air cylinder. The air or steam is transmitted to the cylinder through a globe jointed iron pipe 80 ft. long. This makes the pipe very flexible. The joints are usually 18 in. long. There are thirty of them distributed through all parts of the pipe which require to be flexible. A force of 31,000 lb., if the ground is very hard, can be used to close the jaws of the bucket when it has come down on the ground open. The man in the lever house at the top of the mast turns on the steam or compressed air as soon as the open jaws have settled themselves into the ground and closes them. Each crab takes up from 1½ to 2 cubic yards of dirt and any stones that can be caught between the jaws. The bucket is then raised to any elevation within 50 ft., and swung to any place within a circle of 220 ft., and the load dropped. This is all done in from one and a half to two minutes. The position of the lever man at the top of the mast gives him the full view of the whole surroundings, and he controls all the movements of the crane by levers. The ground to be stripped is from 27 ft. to 30 ft. deep. Three men are required to work the derrick, one lever man, one fireman, and one engineer. In sand they can raise from 2000 to 4000 cubic yards a day, and in cobble a little less. When the mines are first opened in the spring, if during the winter the excavations have been filled in by the winter floods, a space of 60 ft. to 80 ft. square is first stripped and the pay dirt hoisted to the surface, and treated while the stripping goes on on the adjacent ground. The ground from this next stripping is washed into

the pit of the previous one and the tails dumped into the worked-out pit, and so on. The sluices are generally arranged in from two to four strings, as shown in Fig. 35. They are set directly on the pay gravel at a sufficiently high grade to work rapidly.

The water for this purpose is brought from the side hills. The tails and refuse are conducted by a bedrock sluice, 200 to 400 yards long, to a sump, from which they are raised by a pump. To accomplish this, two submerged centrifugal pumps of a capacity of 600 miners' inches of water, or 900 cubic feet per minute, are used, each of which has a discharge pipe 11 in. in diameter. The pumps rest on two 26 in. anti-friction wheels. On the floor above the pumps are two 15 in. Knight's turbines, fed by the main water pipe, with a head of 68 ft. These wheels run the submerged pumps, which raise the sand, gravel, and water to a sluice, from which the sand and gravel are shovelled out. When water is scarce it is roughly filtered and returned again to the mine to be used in sluicing. The pumps raise the sand and gravel 36 ft. vertically as fast as two men can feed it into the sluice. After two years' use they show but little wear. It has never been possible until now, on account of the high water in the fall and winter, to work this mine except from May to November. The high tail banks which have been made by this method, now make it possible to turn the creek so as to work all the year round. There is a very large quantity of ground to which such a process is applicable, but when the vertical height which the tails must be raised is very great, its use becomes extremely difficult. It can be applied to river bed mining where hydraulicking would not be legal, owing to the facility with which the débris can be removed from the bed of the stream allowing water only to pass.

It has always been known that in order to make water carry a heavy load down a sluice it was only necessary to give it a sufficient grade. It has been proved for many years that in order to drive it up a grade it is only necessary to give the water the requisite pressure to make it acquire the velocity which will carry gravel and stones up an incline. The foundations of many bridge piers have been sunk in this way. Within the last few years a great deal of attention had been paid to

the matter, not because it was new but because in the method of forcing the water up by the pressure of air, used for the sinking of the caissons of the piers of celebrated bridges, notably of the Brooklyn Bridge, it was found that large quantities of stones and gravel were carried up with it. Very many interesting experiments were made with regard not only to the quantity that could be delivered but also on the wear of the iron pipes, especially at the change of angle where the detritus fell into the receptacles which were placed to receive them. It was found, amongst other things, that where the change of direction occurred it was necessary to make the pipes of the hardest white cast iron, and that this would resist only a few days.

Whether the inventor of the process which is now being used in California and in Australia for the treatment of placer gravels which lie below water level, knew of these experiments or not, it is difficult to say. The principle which is applied is in both cases exactly the same, the only difference being that in one case the pressure of a column of water, utilised by means of a hydraulic nozzle, is used, and in the other case it was the pressure of air on the surface of the water. After a considerable number of trials, the hydraulic elevators, as they are called, have been developed into a number of patterns. The one, however, which is best known in this country goes under the name of Cranston's hydraulic elevator. Exactly the same principle is used in Australia, and the machine there is known under the name of Perry's hydraulic elevator. Cranston's elevator modified by Hendy has been successfully introduced in California, Oregon, Idaho, Colorado, and Georgia. It has also been advantageously applied in Russia for working shallow places below drainage level. The principle of all the elevators in use is the same, but their construction is a little different. The principle consists in driving the gravel by means of a strong head of water up an incline plane to the head of the sluice, which may be anywhere from 10 ft. to 90 ft. or more above the bedrock of the placer. It has been found that with the best machines constructed up to this time that the gravel can be lifted 10 ft. to 18 ft. vertically for every 100 ft. of water pressure employed. The gravel in its ascent is so comminuted

and broken up by the force of the water and the friction along the pipe and may be raised to such a height, that lines of sluices and other means of catching the gold may be considerably shortened, as great length is rendered unnecessary.

The gravel is generally driven through a pipe or an inclosed sluice which has an inclination of 45 deg. to 65 deg. and a length proportioned to the head of water used. At the base of the inclined flume there is a ground section set as near the bedrock as possible, into which an hydraulic nozzle is introduced delivering a head of water at least 100 ft. and sometimes over 500 ft., which drives the gravel up the inclined flume to the sluices. The gravel and water enter from an open box back of the nozzle. In order to work the gravel two streams are required, one delivering its head upon the gravel bank, from whence the gravel is carried by a ground sluice to the elevator, and the other discharging inside of a box securely fastened to the ground at the base of the inclined flume. The hydraulic elevator runs the water and the gravel up to the sluice above, and consequently can work placers that have no dump, because it can improve the grade by raising the whole of the materials vertically, saves long bedrock tunnels, open cuts, and drains the mines of all the surplus water which comes in through the gravel or the bedrock. When there is room in the pit it may be washed there and the tails only elevated. This method was introduced at Oroville in 1880* by making a flume 400 ft. or 500 ft. long, running from the ground to the river, to deliver the tails. The end piece of this flume descended from the end of the sluice into the ground being worked. It was 50 ft. long and was connected with the sluice in such a way as to admit of its being raised or lowered, and stood at an angle of about 45 deg. The head of water available at that point was 100 in. with 200 ft. pressure. The inclined sluice entered the head of the flume about 1 ft. Where this piece of inclined flume connected with the first box on top of the bank it was covered to prevent the water, gravel, and stones from going over the sides; the rest of it was open like any other flume. The earth was shovelled into it at the bottom and the water turned on. It rushed rapidly up the flume, carrying the gravel with it, and

* Prod. Gold and Silver in the U.S., 1880, p. 15.

after passing through the first section fell into the second sluice on the bank above to be carried off, the same as in any other sluice. A 60 lb. or 70 lb. rock under these circumstances was carried up the incline into the sluice beyond with perfect ease. The drainage water from the river, which amounts to about 75 in. to 100 in., does no damage because it is constantly carried up the incline by the water. This slanting flume was made of wood lined on the inside with blocks, so that the available space on the inside was 1 ft. wide by about 2 ft. deep. No part of this flume wore out faster than the ordinary flume except the top of the incline, where the gravel and boulders struck it before they fell into the sluice.

The machines are built of any capacity and will handle any head of water ordinarily used in hydraulic mining. They are built for the purpose of doing as much work as possible, so that the ground should be arranged to pipe all the gravel into the entrance which can possibly be carried up the pipe to the sluices above without choking it. Fig. 37 shows a view of its application at one of the mines, and Fig. 38 a section through the lower part of the elevator, the sections being numbered for convenient reference. The amount of material which can be forced through these machines is only limited by the volume and the pressure of the water used and the carrying capacity of the pipe, but as it costs just as much time and money to raise water through the elevator as gravel, it should be arranged to carry as much solid material as it possibly can.

It consists of an iron pipe* or closed flume, which is funnel-shaped at its lower end, to admit the nozzle and to allow the gravel and sand to pass in, which is placed upon the bottom of the mine in a pit, from which it inclines upwards, generally at an angle between 45 deg. and 60 deg. In front of the elevator pipe above is the ordinary sluice at any convenient grade. Behind it below is a box made of heavy plank, into which a hydraulic nozzle, 1 in. to 6 in. in diameter, Fig. 38, is introduced. The nozzle and butt 7 and 8 are inclined in the direction in which the gravel is to be moved, and is permanently fixed in a cast-iron fitting 9. The pipe

* For some of this information I am indebted to Mr. J. Hendy, of San Francisco.

1, 2, 3, which conducts the water, has, however, a movement in the joint 4, 5. In front of the nozzle is another fitting of cast-iron, the sluice leading to which is provided with strong iron bars generally 3 in. apart, forming a grating which serves the purpose of

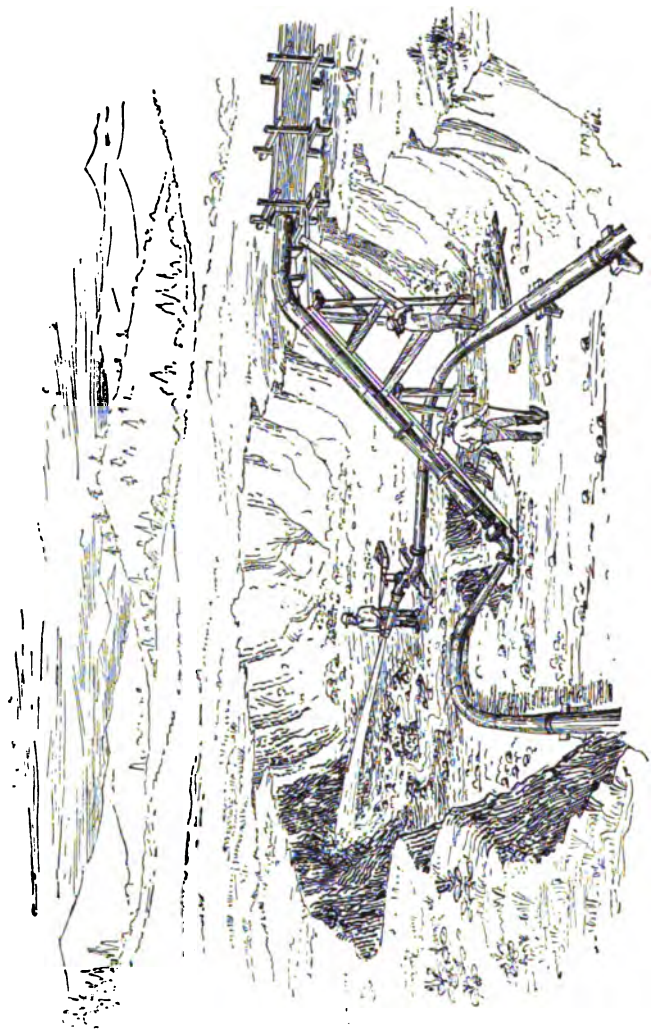


FIG. 37. HYDRAULIC ELEVATOR.

preventing large stones or pieces of wood or any other material that should not pass into the flume from entering at that point. The gravel is washed from the bank with an ordinary nozzle and directed towards this grating. It passes into a short

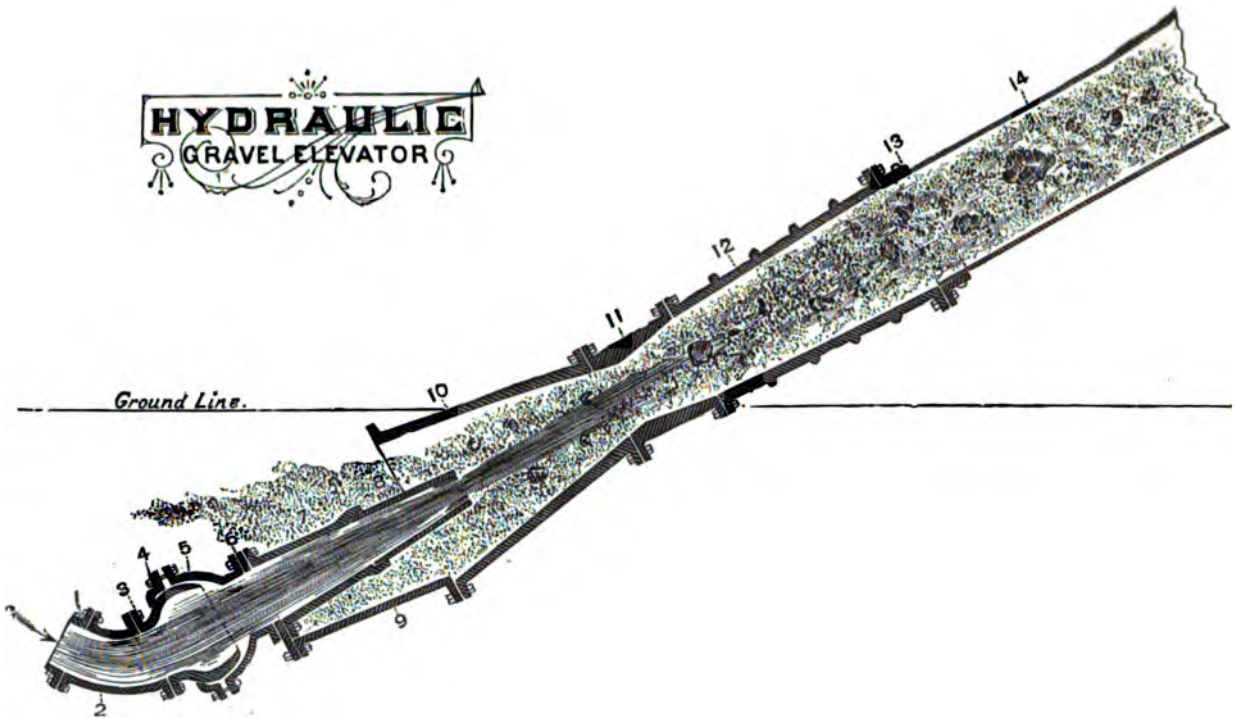


FIG. 38. DETAILS OF THE HYDRAULIC GRAVEL ELEVATOR.

1. Water pipe flange; 2. Half elbow; 3, 4, and 5. Ball joint; 6. Flange between ball joint and butt; 7. Hydraulic butt; 8. Nozzle; 9. Elevator ground section; 10. Elevator entrance section; 11. Elevator throat section; 12. Elevator outer section; 13. Outer pipe flange; 14. Elevator upraise pipe.

sluice from which it is forced by the pressure of the water introduced by the nozzle up the incline flume into an ordinary sluice at the top of the bank, from which it is treated in the usual way. The force of the water and the motion of the gravel as it passes up this short incline, which may be anywhere from 25 ft. to 100 ft. in length, is such that the gravel is thoroughly disintegrated, and the consequence is that the upper sluice may be comparatively short. The undercurrents are placed in it at short distances, and are introduced as soon as the boiling which is produced by such a powerful discharge has ceased. The first undercurrent may be placed within a few feet of the head of the sluice and catches so much of the gold that it is practical to clean it up every day. By properly arranging the depth to which the elevator is put, which should always be excavated out of the bedrock at the lowest possible point, the ground is worked into a funnel shape, so that in almost every case a place may be selected in which a pipe may be fixed once for all and need not change its position. The greater the pressure, the greater the volume of the water used will be, and the larger the quantity of gravel that can be raised at any one time, and consequently the sluice boxes must be proportionately large for the increase of pressure. The size of the giants to be used in every case will be determined by the nature of the ground, whether it can be easily piped or is hard and compact.

To operate an elevator, a shaft must first be sunk from the surface of the gravel to the bedrock, of such size as may be necessary for the convenient working of the ground, and located so that it may be permanently fixed for subsequent operations. The point should, if it is possible, be selected so as to be at the lowest edge of the property, in order that the work may commence from the lowest part and gradually work up towards the highest. By this means not only can the elevator be placed in such a way as from the first to discharge all of the tails, but all the ground in front may be worked down towards the bedrock pit. The shaft having been sunk, the bedrock should be excavated to the depth of not less than 5 ft., for the purpose of establishing the lower part of the elevator firmly in it. The dimensions of these excavations for different sized elevators are given on the

Table, p. 305. It should always be large enough to admit of building into it a box, made either of wood or cast iron, in case the bedrock should subsequently require supports. This box must be securely fixed on string pieces placed crosswise on the bottom. Next to the elevator, back of and below the point at which the sections of the pipe for the entrance of gravel are to be placed, a water lifter should be set, in case the flow of the bedrock water is large. Such an excavation should be at least 2 ft. deeper than the bottom of the elevator chamber and connected with it. This water lifter, Fig. 39, greatly facilitates the work of the elevator. It is used as an hydraulic pump in relieving the elevator chamber from any accumulation of water,

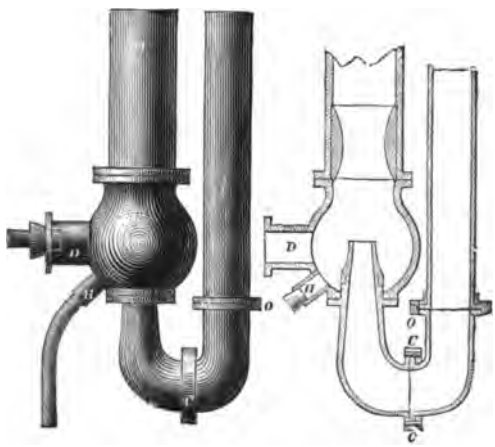


FIG. 39. HYDRAULIC WATER LIFTER.

so that access can be had to any part of the elevator in the event of its stoppage for repairs. As seen by Fig. 39, it is constructed on exactly the same principle as the elevator. It is made in two sizes, Nos. 1 and 2, which weigh, respectively, 600 lb. and 2000 lb. When the bedrock is soft and the flow is not very excessive, an irregular pit may be piped out, of sufficient depth to admit of placing both the elevator and the water lifter in it, that the expense of a timbered chamber or iron box may be avoided. This has been the case in the works at Mamouth Bar on the American River, and those of the Miocene Mining Company, near Oroville, Butte County, California. At N. Bloomfield the excavation is made in the bedrock.

When the elevator excavation is completed, the end sections of the elevator are firmly set on the bottom of the excavation on proper string pieces, the bedrock water being drained from it by the water lifter, which should be previously placed in position. The throat and upper sections are then bolted together in their regular order, as shown in Figs. 37 and 38. The ball joint 4 and 5, Fig. 38, and elevator butt 7 and nozzle 8 are then fixed and adjusted, and the outer sections so set that the incline of the pipe shall be at any required angle. The elevator pipe 12 to 14, Fig. 38, should be supported upon strong bents, Fig. 37, placed on good foundations at sufficient distances apart to be quite sure of their solidity. The Table, pp. 303-305, gives the sizes of all the different parts which are required for the proper setting up of such a machine.

The dimensions of the outer flume are to be governed by the pressure of water which can be had. They must always be larger when the pressure is greater, as the quantity of earth and gravel that can be raised and discharged into the outer flume will be increased with the pressure. The top of the end section of the upper flume should be made of plates of iron or steel, 1½ in. to 2 in. in thickness, securely fastened to it, so as to serve as a shield against the material brought up by the pipe, to change its direction and to prevent its scattering. It must be made of very hard material, for it wears very rapidly, and must be arranged so as to be replaced from time to time. Sometimes it is lined with 18 in. blocks. The line of the upper flumes should have such a descending grade as the contour of the outlying ground will admit. They should be as follows:

First box or 12 ft.	2 in.
Second ,, ,, ...	3 ,,
Third ,, ,, ...	4 ,,
Fourth ,, ,, ...	5 ,,

and the remainder of line 6 in. to the box.

They are constructed as is usual in hydraulic mines. As soon as the elevator is set and the uprise pipe connected, the distributing pipe leading from the main pipe line is flanged on at 1, Fig. 38, the giants are placed in position and connected, and grizzlies arranged to prevent the ingress of

boulders or useless material, which might choke or interfere with the operation of the machine. Before the gravel can be piped into the bedrock chamber, when the ground has been sufficiently opened, the bedrock feed flumes must be put in position, which, like those used in hydraulic mining, may be so placed as to catch any pieces of gold that may be sufficiently free to be caught before entering the elevator. The dimensions of these flumes are given on p. 305. Their grade should not be less than 4 in. to each box of 12 ft., but it is better to have them from 5 in. to 7 in.

The sizes of the giants to be used with the different numbers of elevators are also given upon p. 305. The volumes of water to be distributed to them for bank piping and bedrock sluicing are dependent upon the character of the formation of the deposit to be mined. The sizes of the water lifters to be used are given upon p. 305, and are dependent upon the volume of the water which may accumulate from seapage or from the bedrock. The greater this is, the larger the size to be used. In some cases there is so little water that its use may be dispensed with. The sizes of the main pipes are given below and upon p. 304. The water is usually supplied from distributors like those used in hydraulic mining. These carry the water both to the elevator, water lifter, and giants, and are put up exactly like those previously described. Sometimes the main pipe line is laid directly to the elevator, and the water necessary for the lifter and giants is taken out from branches just above it. The pipe should be made of sheets of varied strength, according to the pressure to be resisted. These sheets should be larger and lighter near the pressure box or outlet from the ditch or reservoir, and smaller near the point where they are to be used. The dimensions of these pipes at Yreka are given in the Table below.

Length of Main Pipe Line (from Reservoir to Distributor).

500 ft.	... ...	22 in. in diameter	No. 16 iron.
800 ,,	... ...	22 ,, ,, ,, ...	No. 14 ,,
500 ,,	... ...	22 ,, ,, ,, ...	No. 12 ,,

Distributing Pipe Line (from Distributor).

To elevator	... 500 ft.	... 15 in. in diameter	... No. 14 iron.
,, water lifter	500 ,,	... 11 ,, ,, ,,	... No. 14 ,,
,, giants	... 1000 ,,	... 11 ,, ,, ,,	... No. 14 ,,

The bedrock flumes to the elevator are 3 ft. wide, 3 ft. deep, with a grade of 4 in. to each 12 ft. The earth, gravel, sand, water, &c., were elevated to a height of 47 ft.; the upraise pipe had an inclination of 42 deg. The outer flume made up of the sluice boxes is 4 ft. wide, 3 ft. deep. The grade of the first box was 1½ in., that of the second 2 in., that of the third 3 in., and the rest 4 in. in 12 ft. Of the 900 in. of water used 300 in. were taken for the giant and 600 in. for the elevator.*

It will be observed that the length of the main and distributing pipe lines are necessarily controlled by the distance between the point of water supply and the deposit to be worked, and the cost of piping is dependent upon that factor.

It is claimed for this process, and has been proved to be true in a number of instances, that, when a pressure of 100 ft. to 300 ft. of water can be had, gravel which lies from 20 ft. to 50 ft. below the surface can be as profitably worked as through an open cut or a tunnel.

The following Table gives the size of the pipes and the vertical distance that the various sized elevators can raise gravel for a given amount of water used. They are the result of a considerable practical use of the machine. The amounts include not only the volume of the water required for the elevator itself but also that necessary for bank piping and bedrock sluicing. The greater the pressure of the water the larger the quantity of material that can be raised.

	Vertical Pressure of Water in Feet.	Volume in Miners' Inches.	Size of Uprise Pipes in Inches.	Vertical Rise.
No. 1 .	150	150 to 300*	12	18
No. 2 .	150	200 ,, 400	16	18
No. 1 .	300	300 ,, 500	12	45
No. 2 .	300	400 ,, 600	16	45
No. 3 .	300	500 ,, 1000	20	45
No. 1 .	400	400 ,, 500	12	60
No. 2 .	400	500 ,, 700	16	60
No. 3 .	400	800 ,, 1200	20	60

The following Table gives the sizes of the different parts of the apparatus:

* From one-half to two-thirds of the water is used by the elevators, the rest is used by the giants for piping the gravel to the bedrock sluices.

——	No. 1.	No. 2.	No. 3.
Size of uprise pipe in inches	12	16	20
Width of outer sluice boxes in inches . . .	20 to 24	24 to 36	36 to 48
Width of bedrock feed sluices in inches . .	36	48	60
Sizes of giants to be used.	1 No. 1 or No 2	2 No. 2	1 No. 2 and 1 No. 3
Water lifters to be used .	No. 1	No. 2	No. 2
Size of main pipes in inches	15	20 to 22	22 to 24
Size of pipes to elevators in inches . . .	9	11	15
Size of pipes to giants .	7	9	11*
To water lifters . .	9	11	11
Dimensions of the elevator chambers on the bedrock in feet in the clear	6 by 12	7 by 16	7 by 16

* Same size to No. 4 giant.

The elevator should be set as at N. Bloomfield, at least 5 ft. deep in the bedrock in a place prepared for it, which is floored with heavy timbers. The lifter should be placed upon one side of the box at least 2 ft. lower than the elevator, and should be used as a pump for the light dirt and gravel. The apparatus should be set on bed pieces placed at right angles to the direction of the flume, and everything should be securely bolted to the bedrock.

There are a large number of these elevators in use in California at the present time, dealing with placers which are quite shallow. Experiments are being made with it at N. Bloomfield to test the question whether the deep placer gravels can be carried up high enough to impound the tails behind dams. The amount of water generally used is between 350 in. and 850 in. As the amount of water is large and it is in some places desirable to economise it, the water used for elevation may be caught and used over again for the purpose of cutting down the bank and of forcing the gravel into the mouth of machine. For this purpose a tank is erected on the side of the sluice at such a distance from the discharge as to allow the water to become somewhat settled. The side of the flume next the settling tank is cut away, but is arranged with bars which form a grating to prevent anything large passing through into the tank, and the surface of the water in this way passes over into the

reservoir. The water is then led by pipes to the top of the bank, and the force is generally quite sufficient to cut down the bank and to force the gravel into the flume.

The water lifter should be set a little distance in front of the feed box. The commencement of the supply sluice should be set outside the nozzle box and at some little distance from the grizzly. The discharge spout is set at an angle of 45 deg. or more. The rise of the inclined flume, if wood is used instead of iron, is 12 in. in 17 in. The top is made of 3-in. plank. The bottom and sides should be spiked, but the top should be clamped on. The top of the uprise pipe should be curved, and should be protected with iron to prevent too rapid wear. In working a number of machines, forks in the pipes are found to work better than distributors, but in any case each supply pipe must have its own water gate.

At the State Mine, near Pino, on the Central Pacific Railroad, a 3-in. nozzle delivers 300 in.* of water with a pressure of 120 ft. The gravel is raised vertically 12 ft. The sluices are inclined 3 in. to 6 in. in 12 ft., which gives 288 ft. of sluice line of steep grade. At the Barron Claim, near Columbia, Tuolumne County, the gravel is raised 30½ ft. with 350 in. of water.

At the mines of the Yreka Creek Gold Mining Company, in Siskiyou County, California, a No. 2 16-in. elevator of the improved form was placed in position and operated during a period of nearly three years. It was there found that a volume of 650 miners' inches of water, under a pressure of 266 ft., was sufficient to supply two hydraulic giants for bank washing and bedrock sluicing, as well as to the elevator and its accessory water lifter with all necessary water for their proper operation in raising all earth, sand, gravel, water, &c., continuously piped to it, to a height of 39 ft., and could have elevated the same over 45 ft. with equal facility. At no time during the conduct of operations was the machine overloaded, and the wear of its several parts was inconsiderable. This shows that the practical operation of the hydraulic gravel elevator developes a far greater effective elevating force than the theoretical estimates.

* Prod. Gold and Silver in the U.S., 1880, p. 327.

The plant consisted of:

One No. 2 16-in. elevator.
,, No. 2 water lifter.
,, No. 2 and one No. 3 giant.

These were operated under a vertical pressure of 266 ft., from a reservoir, with a volume of water varying from 450 to 900 miners' inches, as the season varied.

The distribution of water was in the following proportion, when using, say, 900 in.:

To the elevator	500 in.
,, giants	400 ,,

At Quartz Valley, Siskiyou County, California, on hard ground, where the elevator was first used, it took forty-three days to work out a piece of ground 300 ft. by 250 ft. which had an average depth of 18 ft. The bank was washed down with 600 in. of water and went to the elevator through a 30-in. bedrock flume, which had a grade of 5 in. in 12 ft. The water and gravel were raised through a 20-in. elevator pipe without any contraction at the throat. It was set at an angle of 40 deg. The pipe was 42 ft. in length. The vertical rise was 28 ft.. The force used was 1000 in. of water with a head of 230 ft., through a 6-in. nozzle, and emptied into a flume 6 ft. by 3 ft., which had a grade of 1½ in. in 12 ft. With 3500 in. of water running in this sluice it could not carry off the gravel raised by the elevator, so that it had frequently to be stopped, though it took all the water and gravel which could be piped to it. It could easily have handled 1200 cubic yards of water in twenty-four hours. The work was done without any delay from stoppage of the machine, and no repairs, the wear on the elevator being almost nothing.

The elevator pipe in the year 1887 was made at the lower end of four cast-iron sections of the same diameter, and 18 in. long, which were flanged and bolted together. Above this it is made of twelve rivetted wrought iron pipes in sections, each 3 ft. long, sleeved together. The lower side of the pipe is lined throughout with a cast-iron lining one inch thick on the bottom, tapering to half an inch on the sides; and on the upper side with No. 12 sheet iron. The entrance section flares to 30 in. The nozzle,

which is 6 in. in diameter, is placed just inside of the ground section, and about a foot back of the first cast-iron section, and in the exact centre of the cast-iron pipe. On the bottom of the entrance section, in the centre of it and within half an inch of the stream from the nozzle, a ridge of cast iron is bolted called a "pig," whose object is to prevent the rocks and gravel from wearing out the lower side of the first section of the cast-iron pipe. The nozzle itself is protected by iron bars, which run from the blocks in the bedrock sluice to and around the nozzle, and are set slightly higher than its upper edge. On the top box at the end of the sluice into which the elevator discharges, a casting called a "concave" is set to protect the top of the box and to direct the stream towards the upper sluice, which is 6 ft. wide, and has a grade of 1½ in. to 12 ft. Extra water is introduced at its head to enable it to carry off all the elevated material. The bedrock sluice is paved with blocks. It is 30 in. wide, and has a grade of 5 in. in 12 ft. The wear and tear of the elevator, owing to the softness of the rock and the light pressure at which it is worked, is quite small. It raises about 1000 cubic yards a day 28 ft., and uses for this purpose 900 in. of water under a pressure of 230 ft. at the elevating nozzle. The two monitors washing down the dirt are 3½ in. and 4 in. in diameter. They use 500 in. of water under a pressure of 150 ft. The pipe was at first used without any contraction at the bottom, but the experiment was so successful that a throat section has been recently put in, and the vertical distance to which the tails are elevated increased to 39 ft.

This method is also used at Mammouth Bar on the middle fork of the American River near Colfax. This claim was worked in the early days, but had to be abandoned on account of the quicksands. It had even been drifted within a few years, and the timbers of the shafts and drifts are now being recovered. The plant consists of three elevators and four pumps, which raise both the water and the tails from the sluices, and discharge them into the river which runs against the opposite bluff. Two of the elevators have one pump each, and the other has two. The method of working is to completely empty one pit in the spring, and then fill it with the tails and poor dirt of the next pit. The

nozzle used is 3½ in. in diameter. The trumpet is 7 in. The nozzle is secured in the pipe by a plate into which it is screwed. It projects to the upper edge of the opening for the gravel, on the sides of which there is a flange 4 in. deep. The opening is 12 in. across at the bottom. The piece containing the opening is 3 ft. long and contracts to 6 in. In 13 in. it widens to 13 in. and then reduces to 12 in. This piece is flanged, and to it a pipe 4 ft. long is bolted. As all the wear is at this point, and as besides the pipe is liable to crack here, it is lagged with hard wood, which is kept in place with iron bands, which are shrunk on. The pressure of the water is 460 ft. This pressure raises two tons of material a minute to the height of 64 ft. The lowest angle of the discharge pipe that has been used is 46 deg. The highest is 60 deg. The nearer vertical the pipe has been put the better it works. They have never placed it vertical, as there has been no place where the woodwork could be supported with the pipe in a vertical position. No grizzlies are used on either the elevator or the water lifter. The bottom of the pit is worked like an ordinary placer, with lines of sluices with longitudinal riffles strapped with iron. The tails are run into a tunnel made of the old timbers of the drift mine, and are discharged at the foot of the elevators. A man stands at the front of each of them and rakes out the stones which he judges are too large to be allowed to pass up the pipe. The roof of the box into which the gravel discharges on the top of the bank, is lined with 8 in. of oak, which is fastened by iron and wears a very long time. In the pit all the large stones are moved by a derrick run by a hurdy-gurdy wheel. All the gold is washed in the pit. The tails only are elevated. It is possible to work all the year round but it has not been found profitable ever to do so, as in cold weather the hands and feet of the men become so numb from constant contact with cold water that it is impossible to do the work thoroughly. When the pit is full of water in the spring, the preliminary work is done from a large boat.

This same method is used at North Bloomfield. The injunction of the Courts has prevented very much working for the last two or three years in California by the ordinary methods of hydraulic mining. Some of the ditches are still kept in order,

but their number is small and the amount of water running in them is very much reduced. The Courts now compel the mines to impound all their débris, and to do this it is necessary either to work the upper and poorer benches only, or to elevate the tails from the rich lower benches so that they can be impounded behind brush dams. This necessitates at North Bloomfield the raising of all the débris 96.6 ft., which is done by a pressure of 530 ft. of water, or about 20 ft. for every 100 ft. of pressure. There is no doubt that 100 ft. could be reached with about the same pressure. To accomplish the work sluices are prepared on the lower and rich benches in the bedrock of the old workings with grade enough to carry the gravel. The main sluice in this case is 5 ft. wide and 30 in. deep. It is lined on the bottom with 18-in. blocks and on the sides with 3-in. plank. This reduces the section of the sluice on the inside to 4 ft. 6 in. in width and 15½ in. in height. It is set at a grade of 3½ per cent., or 5 in. in 12 ft. This ground sluice is 380 ft. long. At its end is a grizzly and an undercurrent. The grizzly is made of 1½ in. round iron bars, set 3 ft. apart at the top and 4 ft. at the bottom. They are set so far apart because it would not answer to have all the stones taken out of the sand, for they prevent the gravel from packing, which it otherwise would do. This grizzly is 5 ft. long. It is set with all the angle that the grade allows, which is only about 30 deg., and is much too low. It has a fall of 21 in. in 5 ft., and when the sluice is running full requires the work of two men to keep it clear. Just under the sluice is the drop-box, which allows the whole of the water and small stones and fine material from the sluice above to pass into a sluice below of the same size as the one above. No water ever goes over the grizzly unless stones get wedged into the grating. When it is entirely free, all the water and gravel passes into the sluice beneath. This under sluice is 50 ft. long. At its end it is narrowed to 3 ft. in width. To prevent the wear the last 5 ft. of it is paved with iron grizzly bars, which are 3 in. wide and 1 in. thick, and through these is the aperture for the nozzle below. This nozzle is fixed in the bottom of the sluice and does not project. This is easily done, as the bedrock has been excavated some 5 ft. below the lower sluice. The nozzle is firmly fixed in this excavation

exactly in the prolongation of the centre of the uprise pipe. It was found when this sluice was paved that when the water was turned off from the pipe, the pressure of the falling column of water tore out the blocks, and that when the nozzle below projected through the bottom of the sluice, it was liable to be turned to one side, so as to injure the uprise pipes when the water was turned on again. It is, therefore, kept below the level of the sluice but in the centre of the pipe. The water that washes the dirt has a pressure of 440 ft. It goes through a 13-in. pipe and is discharged through a 5-in. nozzle. It has a Perkins deflector, which is simply a pipe 12 in. long, on the back of which a ring is made to swivel on two rivets. This ring is a quarter of an inch larger than the deflector pipe. At right angles to the rivets are two set screws, which allow it to turn, and these fasten it to the end of the nozzle. Fastened to this deflector pipe is an iron rod ½ in. round and 3 ft. long, which fits into a seat prepared for it on the upper side of the pipe. By raising or lowering this handle the nozzle is depressed or raised, and by turning it to one side or the other moved either to the right or the left hand. The nozzle, which is placed below the flume, is attached to a 30-in. pipe, reduced to 22 in. It is 6½ in. in diameter, and has water at a head of 530 ft. The nozzle is set 12 in. from the opening in the pipe, and on a level with the bottom of the feed sluice. The mouthpiece of the pipe is trumpet-shaped, and is 27 in. in diameter, tapering to 18 in. It is 3 ft. long. The next piece tapers from 18 in. to 14 in. and then opens to 18 in. This piece is 2 ft. long. It is made of sheet iron lined with steel, and wears out rapidly, especially when the pipe is not doing full duty. The next piece goes from 18 in. to 22 in. in 18 ft., but this is not long enough. It should take 40 ft. to go from 18 in. to 22 in. The pipe stands at an angle of 60 deg., when first put up it stood at 45 deg., but did not work well. The box at the head of the pipe is called "the cottage." It is 5 ft. wide, 7 ft. high, and 12 ft. long. For the last 5 ft. the roof descends by a slant to about 1 ft. above the flume, and from thereon runs open. This flume, which is 350 ft. long, has a 2 per cent. grade for 200 ft., until it joins the flume of the upper workings, which is 470 ft. long with a 3 per cent. grade. From there it has 8½ per

cent. grade for 150 ft., and from there for 860 ft. 2 per cent., and from there to the brush dam 3 per cent. The upper sluice from the elevator to the brush dam has a length of 1690 ft. The total length of the upper and lower sluices is 2070 ft.

The method of working the elevator is to let on the water from the lower nozzle so as to completely fill the pipe with water only, and create the strong inward pressure which is necessary. When the pipe is full of water, the gravel is turned on to the feed sluice. The grizzly separates the large stones, and all the monitor can pipe into the sluices, including the rocks which pass the grizzly, is at once sucked up the pipe, and keeps the feed sluice constantly at the same level. All the wear is in the first 20 ft. of the pipe. There is practically none beyond, unless the nozzle stream is deflected, and then the iron cuts through very rapidly. It works without much noise. and works best when it is working at its full capacity, or when there appears to be a little air going up with the water. The water above is full of air bubbles, but the inside of the cottage shows comparatively little wear on the blocks, and this is quite even. The cottage should be lined, so as to have it leak as little as possible. The whole pipe is made of the ordinary iron used in making pipes, except the lower part where the wear is, which is lined with steel. The length of the elevator on the slope is 103 ft.; the vertical height is 96.6 ft. 2400 cubic yards of gravel were piped from the bank by the 5-in. nozzle and washed to the elevator, with the use of 800 in. of water in twenty-four hours. The whole of this water and gravel was raised 96.6 ft. vertically with 1300 in. of water with a 530 ft. head, or a pressure of 210 lb. to the square inch, which was discharged through the elevator by the 6-in. nozzle.

The total labour required for the work of the elevator is one man at the monitor, who directs the stream and keeps the upper part of the sluice from getting clogged with large stones, and two men, one on each side of the grizzly, to keep the stones moving and remove those wedged in between the bars; only one would be required here if the grizzly could have been set at a higher angle. Most of the gold is caught in the bedrock sluice before the sand reaches the grizzly. The 50 ft. from the grizzly to the nozzle rarely pays to clean up. After two or three clean-ups it

gave only 12 lb. of amalgam. The bedrock sluice must be cleaned up as often as the blocks wear out in order to replace them. They cannot be allowed to wear down to 3 in., as they would then cut into the strips 3 in. by 1½ in. which separate the blocks. The blocks so worn are often used for the side linings. The blocks wear very rapidly on a heavy grade, but on one of 2 per cent. they have lasted eighteen months. The upper sluices are cleaned once a month. When working full by the hydraulic process a flask of mercury a day was put in and distributed through the sluice. Now they use only one flask a week. They used to clean up $1000 a day. The loss in mercury is 20 per cent.

The water which is used in the elevator is measured by having a reservoir 25 ft. to 30 ft. square, deep enough and large enough to prevent any wave motion on the surface of the water from the flowing of the water into it. In the front of this reservoir is a vertical board dam with sixteen openings 12 in. long and 12¾ in. wide, with a 6-in. pressure from the top, the water being kept just high enough to keep the top bar of the dam wet but not to allow it to flow over. This size of opening and pressure gives 2000 in., Bloomfield measurement, for each opening. Each opening has a gate, which fits in front of it and is kept there by the pressure of the water. There are also half and quarter gates, but it has not been considered desirable to use smaller ones than these. They are put in place and the amount of water flowing is determined by the number of gates and parts of gates open which are necessary to keep a full head of water of 6 in. over the top of the opening. To ascertain the quantity of water, the number of gates and parts of gates opened is multiplied by 2000 in.

The elevator can be constructed of any capacity, and will supply any amount of water generally used in hydraulic mining. Whenever a pressure can be had, say of 100 ft., 200 ft., or 300 ft., a placer which is from 10 ft. to 45 ft. deep can be worked to advantage. The machines lift 10 ft. to 18 ft. for every 100 ft. of water pressure used to drive them. High pressures work better than low. When 200 ft. of pressure can be obtained to lift 10 ft. high, it does not take quite half as much water to drive the gravel, as to lift 10 ft. high with 100 ft. of pressure, that is,

200 ft. of pressure will lift 20 ft. high better than 100 ft. of pressure will lift 10 ft. These machines require from one-half to one-third of the water to drive them, leaving one-half to two-thirds of the water to be used in the mine to cut down the bank with the nozzles. It will thus be seen that whenever there is sufficient force exerted by the water discharged from the nozzle of the machine to drive it, that earth, sand, gravel, and rocks entering the machine must of necessity be carried along by and discharged with the water at the top. The gravel travels through the sluice at a rapid rate by the direct action of the force of the water from the nozzle. As the whole of the gravel comes in contact with the stream of water it serves to dissolve it and to break it up. It so thoroughly washes and dissolves it, in its passage through the inclined pipe, that long sluices for the subsequent washing, when the pit is not used for that purpose, are not necessary. The greatest amount of wear is upon the lower part of the ground section and the upper parts of the curves at the top of the machine. This wear is upon one side only, where the pipe is not vertical, and consequently the ground section and curves are made in halves, an upper and a lower one, running lengthwise through the machine, with a flange on each side, so that the two halves can be bolted together, and in case of repair the half needing it can be replaced. A vertical pipe has not been used, but the more the angle has approached it the better the machine has worked, and the less has been the wear. The lower half of the elevator and the upper half of the curve is made of chilled iron from ½ in. to 2 in. in thickness. This upper part of the discharge pipe is rapidly worn away by the abrasion of the sand and water, and the repair piece is made movable for convenience. In the experience at the Brooklyn Bridge, where the work was done by compressed air, these repair pieces, constructed of the hardest white iron made from Franklinite ore, did not last more than from three to four days. The force of the water was sometimes 36 lb. to the square inch, and pebbles were frequently thrown with such force, when they were accidentally discharged over the side, as to go several hundred feet upward. In one case a small pebble was shot into

the fleshy part of a man's arm and penetrated it to a considerable depth. There were two cases of persons shot by pebbles thrown horizontally, the protecting iron section having been cut away by abrasion without its having been noticed. In the elevators used in placer mining no such accident is likely to happen. The iron does not wear so rapidly. This wear would be almost entirely avoided if the elevator pipe was made vertical, while the efficiency of the machine would be insured.

The object secured by this invention is a grade for the sluices where the natural surface of the country does not give it; otherwise the method is exactly the same as hydraulic mining. When the washing of the gravel is done in the upper sluice, the gold is for the most part caught in the box at the head of the sluice. The end boxes of this sluice, in that case, pay to clear up only at long intervals.

The principle of this elevator has been used in working the tails and the pay ground underneath them, which has been covered by them from previous workings. The tails are simply worked over by this method, and as the force of the water is so great that the sluice need not be long the dump is simply moved a few hundred feet and left. When the tails have been worked off, the virgin ground below them can be treated. The expense of this method of treatment is small, and where water is abundant and cheap comparatively poor ground can be worked by it. The larger part of the gold is caught. A large part of the water when it is scarce can be used over again. It is consequently very cheap. It seems as if here was a solution of the very difficult débris problem, and so it ought to be for very small and shallow placers, but that is not the problem to be solved. The problem is the working of deep placers, for which this process so far as it has been tried has not proved to be successful on account of the very frequent stoppages for repairs, and the fact that more than double the amount of water necessary for hydraulic mining has to be used, and the solution of that problem is yet to be made. It will, however, be possible for the mines to raise the gravel, wash it, and then return it to the worked-out grounds for a dump, and thus avoid tailing the gravel into the rivers. To what extent this can be done on a very large scale

on such low grade gravel as is worked on hydraulic mining remains to be seen.

Seam Diggings.

Seam diggings is a name given to a purely local phenomena which occurs in the central part of California. The seams consist of decomposed rock, filled with irregular veins of quartz, which contain both gold and sulphides. They run in every possible direction, do not have any uniform dip or angle, and do not appear to unite in depth like an ordinary vein, although some of them are large enough to be fairly called so. This kind of rock has for many years been worked by the hydraulic process, but within a few years, owing to the more economical construction and operation of mills, it has been possible to mine, crush, and concentrate and then when it contains sufficiently rich pyrites to chlorinate it. The seams are found in El Dorado County, California, and elsewhere, in a very rotten rock, a number of them usually lying side by side and parallel to each other, but sometimes interlaced. The rock is usually a decomposed slate, which is easily broken out with the crowbar and pick. Only occasionally a small quantity of powder has to be used. Shafts or inclines sunk on such seams generally show that they continue downward, though the rock grows firmer. The veins are usually from ½ in. to 3 in. or 4 in. in thickness, and generally have a north-west strike and a northerly dip. They are sometimes 6 ft. to 8 ft. apart. To work such claims, open cuts are made into the hill. These cuts are from 50 ft. to 150 ft. wide and from 200 ft. to 300 ft. long. In this space there may be 50 or 100 such veins, most of which, especially the smaller ones, are usually very rich. Most of these cuts are washed out by the hydraulic method. The dirt is sent through sluices. The hardest rock is stamped and then washed as usual. Working such rock is naturally profitable and the method of mining is called seam digging.

At Georgia Slide, they were formerly worked by the hydraulic process to the depth of 175 ft. and then by shafts and drifts. The material is now crushed in mills. The seams are not uniform in the quantity of gold they contain and would be much more profitable if water power could be used in the mill. At Spanish Dry Diggings the country rock is slate and a fine grained sand-

stone filled with crystals of pyrites, irregularly traversed by quartz seams. The seam diggings are in a fine silicious conglomerate containing a little feldspar. It is intersected in every direction by pockety veins of quartz, and covers a large area in the centre of the county. The following Table* gives the results of examinations of the rock made by the California Water Company:

—	Working Test.	Yield per Ton.	Remarks.
	lb.	$	
Nagler Claim	35	1.31	
,, ,, ...	35	1.18	Contained 1 per cent. of sulphurets.
Nagler tailings sample (below sluice head) . . .	32	1.40	
Crane's gulch tailings . .	45	.31	Contained 2 per cent. of sulphurets.
,, ,, lode . . .	48	1.01	
Nagler porphyry . . .	32½	17.00	One piece of free gold vitiated the assay.
,, sulphurets . . .	1	325.00	These were concentrates.
Crane's gulch sulphurets .	1	678.00	

Debris Litigation.

What the future of hydraulic mining in California is to be is still a matter of uncertainty. The least that can be said is that the conflict between the agrarian and mining interests has put a serious check on the growth of its prosperity for the present. If the only damage that had been done had been to the lands owned by the farmers, the litigation would have to be called at the least short-sighted, because the farmers knew when they occupied these lands what was being done and what was in store for them; but damage has been done to the navigable rivers as well, and, according to some persons, there was even danger to the bay of San Francisco. However little occasion for fear there may have been of any such result, there was danger that further damage would be done to the rivers, for some had already been done, and as we shall see, the result of the litigation affords no relief from it. Many of the important mines have been closed; others are endeavouring to impound their tails behind dams or are throwing them into side cañons from which they cannot escape into the rivers; others have taken refuge in drift

* Prod. Gold and Silver in the U.S., 1880, p. 345.

mining, so that they work only the lower part instead of the whole deposit including the top gravel. There is no fear that the gold in California will be neglected, but it will cost more to get it than it did before, for a time at least. It is much more likely in the year 1888 that some one will invent a process, or that a process will grow up of itself by which the gold can be economically extracted from the gravels which contain it, than it was in 1852 that the stupendous achievements of the hydraulic mining of to-day would in thirty years grow out of a little ditch, a flour barrel, a cowhide hose, and a tin horn. Some other inventor, who will probably be better known to fame than the miner at Yankee Jim, will probably arrive to give a new method of working, which will as easily override all the impediments which the farmer has to-day put in the way of the miner, as the invention at Yankee Jim overcame the difficulty which the miners were then contending with.

The damage actually done is undoubtedly large. It has been estimated* that since the commencement of the hydraulic mining, to about the year 1876, 162,000,000 cubic yards of gravel have been sluiced into the Yuba River and its tributaries, and that the amount now retained in it or lodged within its cañons will not exceed 12 million cubic yards. 150 million cubic yards have therefore passed the foot-hills and have been deposited in the bottoms of the Yuba, the Feather, and the Sacramento Rivers, the bays of Suisun san Pablo, and of San Francisco. The Excelsior Mining Company admit that they alone have sluiced 18,600,000 cubic yards of gravel into the Yuba River. This enormous mass would cover 150 acres 581 ft. in depth, and would cover 145 square miles of land one foot deep in débris, which would destroy it for agricultural purposes and render it unfit for the growth of anything except willows and similar plants. Others estimate that up to the present time there have been deposited in the Yuba River and its tributaries 750,000,000 to 800,000,000† or more cubic yards, distributed for hundreds of miles along its channels, and covering 40 to 50 square miles to a depth of 10 ft. to 150 ft. The surveys of the State engineer of California have demonstrated that 18,000 acres

* Look on Gold, London, 1882, p. 994.

† Prod. Gold and Silver in the U.S., 1884, p. 528.

of bottom land on the Yuba River, which was at one time some of the best in the whole State, have been destroyed by being buried underneath mining débris. It is estimated that fully as much has been destroyed on the Bear River. These lands do not produce a blade of grass. Only willows, which derive their nutriment chiefly from the water underneath the débris gravel, and plants of this nature, grow there. It is estimated that on the Yuba River alone, during the dry season, 17,000 miners' inches of water are moved daily and that each 24 hour miners' inch moves three cubic yards of gravel. This gives a daily movement of 51,000 cubic yards. It is estimated* that half of this is carried down the river to be deposited partly on the bottom lands and partly in the rivers and bays beyond them. This means that 25,000 cubic yards, or 43,750 tons of gravel and sand are turned every day from the mountains into the valleys below. If such a mass of material has to be moved by ordinary means, it would take 110 trains of 40 cars, with a train every thirteen minutes for the twenty-four hours to do it, and this would be correspondingly increased in the wet season, which would cause movement in the sands at the bottom of the river as well as those discharged into it. So stated the grievance appears to be a just one on the part of the farmers, but this is not the whole of the question. For more than twenty-five years the débris of that district has been discharged in this way. The farmers were fully advised of all the facts from the beginning and knew what the condition was when they settled in that territory. They had already diked out the high water from some of the towns before hydraulic mining was invented. Prior to the commencement of this litigation, they did not object to the use of the rivers as débris deposit. The effect of the deposit was just as well known on the Yuba and Feather Rivers and in other parts of California then as now.

That these rivers were subject to great floods, which carried enormous quantities of débris down with them, was known to the Russian exploring expedition, who reported to their government in 1824, that on account of the periodic floods, the valley of the Sacramento was not habitable. This opinion was partially endorsed by Commodore Wilkes in 1842. The great freshet of

* Lock on Gold, London, 1882.

1862 swept out of the rivers the accumulated débris of fifteen years of mining. It is estimated that the total damage done since the commencement will not exceed $1,500,000, so that most of this is offset by the reclamation of swamps and other lands now covered with débris, which, after it has been covered a few years, has no equal for the production of many kinds of fruits, and is perfectly suitable for hops, potatoes, and other vegetables. The miners assert, justly, that there has been no change in the mode of mining, causing any new exigency; that they have spent millions of dollars, which has resulted not only in creating a new industry for the State of California, but that it benefits the whole of the United States by developing its resources of precious metals and making it possible to bring the new use of water power, to the development of new industries, in parts of the State where other power could not have been used, and also of sustaining old and well developed industries, by furnishing them with cheap water power at a time when the production of precious metal by means of steam was commencing to be a real burden, and when the time could be foreseen in certain districts of the State when without the cheap water, the mines would have to close. They also claim that they have furnished cheap water to the farmer for the irrigation of his lands, which has rendered thousands of acres tillable which would otherwise have yielded no crops. The miner also asserts that the decision has already deprived 20,000 white miners of their living, and has frightened away at least 10,000 more, besides undoubtedly affecting 20,000 others, a total of 50,000 persons in all, leaving no work for white men who obey the law, but forcing them to look calmly on while Chinese take possession of their homes and claims, and work in spite of the law. Besides, it must be remembered that the miners purchased their lands from the Government, paying double the price paid for agricultural lands. The lands were sold by the Government with the privilege of mining and with the full knowledge that the gold contained in them could not be extracted except by the hydraulic processes. On the good faith expressed in the title, the miners have expended millions of dollars, founded a great industry, and executed some of the greatest engineering works of modern times, and they ask and ought to have some kind of protection

Both the State and general Government are moving in the matter of this litigation, and it is hoped that some result will be effected which will protect every interest and produce harmony at the same time.

As the product of gold from the hydraulic method is likely to be checked for a number of years by the injunctions recently placed on the leading mines, which will continue in force until dams are built in the side cañons or the injunctions dissolved on an appeal to the Supreme Court of the United States, a history of this litigation will not be without interest. In September, 1876,* a farmer owning land on Bear River, alleging injury to his land by the operations of the miners, commenced an action against thirty of the large mining companies and one hundred or more individual miners who dumped their tailings into Bear River, whence they were washed during high water upon his lands. This suit was lost on a technical point. Other suits were then brought for the same cause by other farmers residing on Feather, Yuba, Bear, and American rivers, and it was judicially determined that certain lands on or adjacent to the navigable waters of the State have been damaged by being covered with slickens or débris, which is left on such land during the stages of high water, being carried in suspension from the operations of hydraulic mines. This culminated in a suit in the United States Circuit Court, brought September, 1882, which was decided against the miners. After this, several companies, the largest producers of this class, united to oppose an action brought by the representative of the Anti-débris Association, asking for an injunction on the ground that the operations of the miners constituted a nuisance. The injunction was granted with a modifying clause, but the decision of the Circuit Court had the effect of reducing the production of gold ten or twelve millions per annum, as none of the companies can work their ground by any other than hydraulic processes. The companies† who defended the suit discharged their men, ordered their waste gates to be thrown open, and their 175 miles of canals and flumes which were constructed at an expenditure of over $700,000, were left to

* Prod. Gold and Silver in the U.S., 1884, p. 526.

† *Ibid.*, 1883, p. 716.

take care of themselves. The Milton Company, having 80 miles of ditches, and the Eureka Lake Company, having 173 miles of ditches, which cost $400,000 and $725,000 respectively, were also stopped. The main damages which will result to this property will be from snow slides that will probably damage some of the flumes.

Finally the State of California, through her Attorney-General, brought an action against the Gold Run Mining Company, of Placer County, to test the question of the right of the company to flow its tailings or débris into Cañon Creek, a mountain stream about 90 miles distant from the nearest navigable river. This, after a long and ably contested trial in the Superior Court, was decided against the Mining Company, and after an appeal the decision was confirmed by the Supreme Court of the State of California in November, 1884.

It was the intention in all of these suits to enjoin every hydraulic mine in the State, and that this object has been partially accomplished is shown by the falling off of the gold product during the years since then. All the great companies have ceased to work in their own name at least, and the smaller ones are gradually giving it up. This must continue until, either by national or State legislation, a modification of the decrees of the California Courts by the Supreme Court of the United States shall find a remedy and determine under what restrictions, if any, hydraulic mining may be lawfully pursued and the gold product of California brought up to its former average.

After the excitement of the contest and the enforcement of the law for several years, the conviction is gradually forcing itself upon the public mind that great injustice has been done, not only to the hydraulic miners, who produced two-thirds of the gold mined in the State, but to the productive industries in general, as making it uncertain what their future is to be. The public conscience has grown uneasy under legislation in favour of a single class, the farmers, and against another class, the miners, but which does not affect them alone, but more or less directly every producer in the United States. It is evident that legislation injuriously affecting so many millions of property cannot remain in a settled condition, and that, as the law cannot afford relief, the

legislatures of the State or of the United States must be applied to for the purpose.

The injunction issued by the Courts did not restrain, however, all hydraulic mines in California from working.* Only those whose outlet for tails was on some of the branches of the Sacramento and San Joaquin rivers, as the American, Bear, Yuba, and Feather rivers, were affected by it. These, however, include the largest and most productive mines in the State, being those located along the main gold belt, extending from Tuolumne north to Plumas County. South of Tuolumne County, the hydraulic deposits, though in places quite rich, are not extensive and have never been worked on a very large scale. In the range of counties extending 100 miles north, there are many extensive and valuable beds. Throughout all this territory, the mines have stopped working, waiting for either some change in the decision of the Courts, or the invention of some machine like the elevators, or the use of some principle, like the impounding of the débris, to set the mines working again; but for the present hydraulic mining is certainly suppressed. It is, however, extending as a branch of mining industry in the north-western counties of the State, where it has not yet been enjoined and probably never will be. All those mines located in Trinity, Humbolt, Siskiyou, and Del North Counties, which discharge into the Calumet River and its tributaries, have not been molested and are not likely to be, since these rivers are not navigable and no rights are likely to be jeopardised by the washing of the gravel. There are here large quantities of auriferous gravels, which are situated in the most favourable conditions for mining, and although operations here are not carried out on so large a scale as they were further south, they are likely to increase both in size and importance, as there is plenty of water and the grounds moderately rich, and there is no débris litigation. The working has been done in a limited way, the gravel being found mostly on the Klamath, Smith, Trinity, and Salmon rivers and their branches. These deposits usually carry one or more channels extremely rich, and the gravel is usually richer throughout than in those where hydraulic mining is now enjoined, and is characterised by the almost entire absence of cement and volcanic débris.

* Mining and Scientific Press, vol. liii., p. 281.

Water is also more abundant and it consequently costs less. There is generally sufficient of it to work eight to ten months of the year without storage, and the working of the mines is not interfered with either by snow or ice. As none of the rivers in the north-western part of the State are navigable, and run for the most part over rocky beds, there will never be any danger from litigation here. The larger part of the farming land lies high above the water, so that there is no fear of interference from the farmers, and it is to this part of the State that attention is being now turned.

The total area affected by the suppression of hydraulic mines* extends from Feather River on the north to the Cosumnes River on the south, a length of 144 miles of auriferous territory on the western slope of the Sierra Nevadas, intermediate between the foot-hills and the high sierra. The average width of this gold-bearing territory is about 48 miles, giving an area of nearly 7000 square miles, all of which may be classed as mineral lands, though only a small portion could be worked by the hydraulic process. Within these limits it is estimated that more than 50,000 men have been thrown out of employment, and a great depreciation of property as well as population has followed.

It is a well-established fact that there are in the Yuba River and its tributaries 750 to 800 or more millions of cubic yards which will find its way down unless checked in some way, as it covers forty or fifty square miles. This is already in the river to the depth of 10 ft. to 150 ft., and has been accumulating from all sources since 1862. The injunction will not stop the difficulty, because this material will slowly but surely find its way down. The rains of the winter of 1884 have developed the fact that although most of the large mines have stopped work, yet the flow of débris down the Yuba River below the mountains is nearly as great as ever. So long as mining continued, the water flowing from the mines was loaded with material to such an extent that it was not capable of transporting much more taken up from the rivers; consequently it passed laden with its ordinary load until it was at comparative rest below, when it dropped its load, not of

* Prod. Gold and Silver in the U.S., 1884, p. 526.

† *Ibid.*, p. 528.

sand but of fine silt. Since the farmers have succeeded in closing most of the important mines, the quantity of water in the rivers has been reduced, as the great mining reservoirs are either entirely emptied or not being drawn upon, and at the same time there is more material in the shape of sand moving than heretofore. It continues to move until it reaches a point where, owing to the grade of the river, the water drops it, and consequently the lower rivers are steadily filling and will continue to fill until the river channels above are cleared to the bedrock. During the winter season, the increased quantity of water augments the flow of heavy material, but there is a point where it will only pass during the high stages of water, which is for a limited period of the year only. The injunction requires that the débris from the hydraulic mines shall be impounded or caught within the side cañons and not discharged into the rivers. If this can be done cheaply hydraulic mining can continue.

The question at the present time is what can possibly be done with the nearly 5000 miles of ditches in the State of California? The aggregate length of the twenty-four which have been constructed on the Sacramento and San Joaquin rivers is 1470 miles. These are known to have cost $10,176,843, and their storage capacity for the dry season is 2,158,300,000 cubic feet.* Over very large areas it has been ascertained that ground containing on the average one cent to the cubic foot would, in large tracts, be exceptionally rich, and that ground containing one mill. to the cubic foot has been worked with profit, while the minimum of profitable working by the drifting method would be about 10 cents per cubic foot, and consequently this last method is in most cases out of the question. The future therefore does not lie in the method of drifting. It has been proposed to raise the gravel from the banks, to wash it and afterwards return it to the old places from which it was taken, but at the present time only a very few such works have been constructed, and it is very doubtful if they can be worked on any such scale as hydraulic mining has been. The question whether or no the gravels can be impounded with security rests solely on the construction of dams of great height and strength in the beds of the

* Prod. Gold and Silver in the U.S., 1884, p. 530.

mountain streams below the mines, and at a suitable and safe distance above the valleys. Such dams have been pronounced practicable by some of the best of engineers. They would settle all the heavy material behind them, where it would be impacted and become as firm as the original beds, but the water which flowed over the crests of these dams or through the escape gates would still carry a quantity of silt in suspension, estimated at about 20 per cent. of that in the mountain streams. The question then becomes one of additional expense in the outlay for the construction, as the dams would have to be raised from time to time as they filled, or others constructed above them. This system has been in operation on a small scale on Slate Creek in Sierra County, and its efficiency has been practically and satisfactorily demonstrated, but it remains to apply it on a very large scale.

There seems to be now no other possible way of mining the gold on the same enormous scale that has heretofore been done than to allow the débris to run into the river, but there is but little doubt, should the Courts refuse in any way to modify the injunction, that in the course of time some other method will be found by which the gravel can be worked.

Notwithstanding the injunction, hydraulic mining is still carried on in the southren tiers of counties, though it is no longer legal. The owners of the claims either find means to evade the injunction or lease the mines to the Chinese, to whom the law is of no consequence, and who evade it in the open day. When the officer of the law goes to stop the working and arrest the persons who are responsible, there is no one on whom to serve his papers and no one appears as the owner. If he arrests one Chinaman it does not stop operations. This one, with his own and the consent of his comrades, cheerfully goes to prison, where he gets his wages paid by his employers, as if he worked, and fares, to him, sumptuously, as the prison fare is much better than any he ever gets while working, and he is perfectly willing to remain in prison so long as he can be paid and fares well. It is useless to fine these people, for the fines cannot be collected, and as useless to imprison them, for that is no punishment. The responsible parties in the case cannot be reached, and the proprietors

of the leases are perfectly willing to bear the expense and the legal annoyance, since they get the mines, with a complete outfit, for almost nothing, and their profits are very large. The law is broken, its officers defied and brought into contempt, and thus what has been estimated the most profitable industry of California is becoming, if it has not already become, the nursery of crime. A possible output of all the legislation is that the hydraulic mines are likely to fall into the hands of Chinamen. This seems at first a simple and a natural solution of the difficult problem of allowing a foreign people to become permanent lawbreakers, and only the moral side of the question is seen by most persons, but there is another and a much more serious one for the State to consider, which is that beside all the loss which must be undergone by the owners of the mines, who lose very valuable investments, amounting to more than one hundred millions of dollars, which is so much dead loss of capital to the State, all the gold which is produced from these mines operated by the Chinese goes immediately out of the country to China, so that, instead of enriching, these people are actually impoverishing the country, as the result of legislation which, to say the least, if it was good law, has been most unfortunate in its results and merits the serious attention of the lawmakers, so that they shall not be educating lawbreakers.

The present outcome of all the litigation is that all the very large mines are stopped; one of the greatest industries of the United States is, for the present at least, broken up. But the relief which the farmer had expected has not been afforded by the injunction, for the amount of débris is not affected by the decision of the Courts, but still moves and will continue to move so long as rain or snow fall or there is water in the streams or débris in the rivers. The farmers have gained their suit, but the effect which made them bring it still continues; they are if anything a little worse off than before, and they can get no further advantage of the miners and must now put up with the inconvenience about which they so bitterly complained, until the débris is all washed out of the rivers or there is no loose earth on the hillside.

CHAPTER III.

DRIFT MINING.

WHEN the auriferous gravels contained in the ancient river channels are covered over to any great depth by the volcanic eruptions of past geological periods, or where the pay gravel is concentrated in a layer, from 2 ft. to 8 ft. thick, directly on or near the bedrock under a very thick deposit of either barren or very poor gravel which could not be profitably mined by the hydraulic process, or where, from scarcity of water or difficulty of getting dump room, owing to lack of fall of the ground below the mine, it would be impracticable to move the whole of a deep placer deposit, the bottom gravel only is mined and the method of working adopted is called drifting, and the claims are called drift claims. The volcanic rock or gravel above the gold deposit cannot be moved, and the ancient channels, if reached at all, must be reached by tunnels run through the side of the rimrock at sufficient depth to reach the deposit and make it possible to take out the pay in the ancient channel. Drifting is never done where hydraulicking is possible. It has come into notice recently on hydraulic claims since the débris litigation, but would never be practised where it is possible to get rid of the whole of the ground.

When the great lava overflow of the Pliocene period swept over the surface of the country, it covered it to many hundred and sometimes fully 1000 ft. deep. This lava flow necessarily filled in the deep channel depressions, and there was enough besides to cover nearly all the elevations as well. The present streams, such as the Feather, Yuba, and the American rivers, with their forks and branches, have gradually eroded great cañons in this volcanic flow, in which they now run 1000 ft., 2000 ft., and 3000 ft. below the summit of the lava-capped hills. Their general course, more particularly in the region of the lava-covered river channels, is

westerly. The erosion first cut away the lava, then the gravel, and finally cut down into the hard slates. This erosion was the principal cause of the formation of the gold-producing river beds and bars which were so extensively worked in the early days of California mining. The discovery of these deep leads is probably owing to the fact that certain miners, being unable to obtain possession of either the placer, river, or bar claims, worked upon the hillsides until they were led underneath the surface of the ground, which they followed into as far as their limited means would allow.

The deposits adapted to drifting are very extensive in California, but their development will take place slowly, as it takes time to explore them and capital to open them up. As a general thing, as they work in the dark, it requires more intelligence to make the preliminary surveys than in any other kind of gold mining. They do not invite speculation on account of the great risk which it is necessary to run in the first opening of the mine. The experience is general that they can be worked to best advantage by associations of companies of practical miners. All the great successes have been achieved by companies of this kind. The gold product from this scource is estimated at between $3,000,000 and $4,000,000 a year, but it will undoubtedly be greatly increased in a few years.

Drift mining is a permanent industry of Placer County, and is extensively carried on in Nevada, Sierra, and Plumas counties, and on a small scale in El Dorado County. The gravel is not always found in exactly the same condition. Some of it is ordinary gravel, easily mined with the pick and the shovel. Other gravels are hard but do not require blasting, and others are so hard and tough as to require to be blasted in the mine and afterwards to be crushed in stamp mills before the gold can be extracted. Such gravels are called cement, and the mills are called cement mills.

It is a matter requiring the greatest judgment of the engineer to settle what it is advisable to do, to successfully open a given piece of ground which he is examining. The probable width of the channel can only be guessed at. Its depth can be ascertained by prospecting. As a general rule, it must be said that a knowledge of the known channels which have been worked out in a

given locality is the best guide to work on. Long experience has shown that the highest slope of a channel will not usually exceed 30 deg. They vary from 40 ft. to 50 ft. up to several hundred in width. They will average about 250 ft., but as a general thing will not exceed 600 ft. to 700 ft. in width.

The method of drifting was formerly employed to a small extent to work the sands of the bars of the rivers, which, as they are loose, were easily mined, but they had to be worked on a small scale, as there was but little material to work upon. The idea was to follow the pay leads under the banks, supporting the excavations by boulders, and working for short distances only, leaving holes rather than drifts, which soon caved. This method, which could not be worked on any system, was called "coyotting." It is not a method of mining, and was never known to capital. Drifting did not become a branch of mining until 1852 or 1853, about the time when the old river channels were discovered. In the early stages of its history, many mistakes were made, which were very disastrous, owing for the most part to the tunnels having been run too high up to open the whole of the ground. Entirely underrating the difficulty of following them, shafts, inclines, and adits were sunk in every direction in the old channels of Sierra, Nevada, Placer, and Tuolumne counties, the large majority of which brought only failure to individuals and discredit on this method of mining. The enterprises were undertaken by the miners themselves, who either worked alone or in associations, the day's work being counted as so much invested capital. The modern methods of exploring ground, to determine the position and depth of the channel, were unknown. No shafts were sunk, the pay streak was followed where it was found. When the company had a number of members they worked until the money gave out, and when it was gone they worked as day labourers somewhere else until money enough had been accumulated to commence work again. As the higher up the hill the tunnel was run the shorter it would be, they worked high to realise quicker, and frequently lost everything, never reaching the pay streak at all. As success depended on chance and not on skill, it is estimated that fully 75 per cent. of these early ventures were failures. When the old

channels became the subject of careful study the mistakes became fewer, the method as certain as others, and it has now become a very profitable branch of mining.

The amount of gravel in which the gold exists in drift mining claims, is not less in extent than that which is worked by hydraulic mining, but the poor upper gravel cannot be extracted with a profit in drifting on account of its position under the volcanic rocks or poor gravel, and they must, therefore, be left and only the rich bedrock section be worked. In many cases, these deposits are found in basins which are formed by the swelling of the bedrock, so that for a considerable time there was a theory in vogue that the gold deposit was formed from lakes instead of rivers. The formation of these basins is owing to the fact that the ancient rivers differed in no respect from the modern ones. They were on a very much larger scale, however. They had the same bars, eddies, rapids and waterfalls, and the same peculiarities of the gold hiding itself away in corners and crevices. There were in these rivers, as in those of to-day, long barren stretches, and pockets or crevices which were of great richness. In the modern streams, such rich or barren places are found from external indications without much difficulty, while in the ancient streams, covered with lavas or heavy beds of poor dirt, they must be sought for almost at haphazard. It often happens in these streams, as in the modern ones, that there was more than one channel—not unfrequently two and sometimes as many as three. They are not of necessity equally wide, but they must not be passed over if they are there, as the expense is nearly the same whether there are one, two, or three The problem of drift mining is to find the "channels," to open them up with the least possible expenditure, and then to work them with the least possible cost. It is not always possible to find any surface indications of an "old channel." Some of the best mines worked at the present time, and nearly all those worked out and abandoned, were found by following the surface placers up to the line of the disappearance of the gravel and gold. At about this point there is generally a "channel opening," and here prospecting must be done. Sometimes a depression in the line or under edge of the country rock indicates an ancient

"channel" and the probable presence of gold, but this cannot always be relied upon. Gravels, however, are not necessarily confined to a definite "channel," as in many cases the ancient streams, like the modern ones, overflowed their beds, carrying gravels and sands with them wherever they did overflow. Nearly all of these sands contain some gold.

Drift mining has been brought prominently before the public by the débris legislation, because it was thought by some persons that it would solve that question, as it attacks the gold on the bed-rock, and less material therefore is washed, only the richest part of the claim being worked. The districts in California where this method is practicable may be said to comprise all the deep placer districts which have not been worked out. They are included within the counties from Mariposa on the south to Shasta and Siskiyou on the north. It is, however, in Nevada and Placer and Sierra counties that the most extensive claims are being worked at the present time. There are considerable areas in Plumas, Butte, El Dorado, Amidor, Calaveras, and Tuolumne counties which are being worked. As is well known, the main axis of the Sierras, including the western slope of those mountains, to a distance of from ten to forty miles from the summit, is granite. From the western edge of the granite, and overlying it, there is an immense body of metamorphic rocks. Of these rocks slate forms the largest part, limestone being the next. In a large area of the western part of Placer, Nevada, and El Dorado counties granite appears on the surface and is the ordinary country rock. The age of the slates and limestones, so far as can be determined, has been settled as Jurassic and Triassic. The quartz mines are contained for the most part in the granite and slate on the line of separation between the two formations. In some localities two distinct sets of ancient river channels have been found, one cutting across the other and so different in every essential characteristic that they were undoubtedly formed at very different periods.

The system is used on a large scale in Sierra County on the Blue Lead, an ancient river which is supposed to have had its source in the high mountains of Plumas and Lassen counties. The channel has a width of 200 ft. to 500 ft. Here the pay gold is

quite coarse, and is confined to a stratum from 2 ft. to 4 ft. from the bedrock. The "Blue Lead," as it is termed, has a grade on its southerly course of from 50 ft. to 175 ft. to the mile. It was opened in many places as early as 1853, and has been continuously worked since that time. The returns from the early workings, comprising a distance of several miles upon the course of the channel, have been estimated from $200 to $300 per linear foot, as nearly as may be determined from the imperfect records of the early operations.

These gravels sometimes cover the whole surface of the country, without any apparent defined channel. There is also the greatest variation in the consistency of the volcanic rocks which cover them. Some of them are extremely hard, others thoroughly decomposed and easily separated. The channels are subject to a certain number of irregularities and disturbances, and are sometimes even temporarily entirely cut off. These breaks are often called by the miners "lava flows," but as a general thing they are owing to the displacements of the bedrock by the weight of the gravel above, or else they are disturbances caused by a movement of the pipe-clay. It is not an uncommon thing for the bedrock also to swell, and in some instances it is extremely difficult to keep the tunnels open on account of it. Occasionally the bedrock appears to rise and then to descend, and in many cases this has undoubtedly been induced by the additional pressure caused by the erosion of large bodies of the ground, leaving great spaces between the ends of the "lava flows," thus causing the bedrock to bulge. When the bulge occurs in two places at some distance apart, it gives the appearance of a basin, and often, when only two sides of it are found, leads the inexperienced miner to suppose that his claim is worked out in that direction.

It is generally true when two slopes of the sides in opposite directions have been found that the rim has been reached and the working is stopped in that direction, and this is the rule always followed except in the case of the accidents to the channel or bedrock already referred to, when the channel has two or more "gutters." In such a case the rimrock must be followed up the incline until the "sand streak" or poor

gravel coming down, gradually cuts out the pay gravel and the bedrock is finally met. Generally, in such an exploitation, if there are several gutters, the inclined rock supposed to be the rim will turn down so as to give the indication that further exploration must be made.

The gold in these deposits varies greatly in size. It occurs from fine float dust up to such nuggets as were found at Bald Mountain, which were from 40 oz. to 60 oz. in weight. Most of it, however, lies in a few feet of the gravel on the bedrock, while the whole of the upper gravel is too poor to work. Some few of the present hydraulic mines were originally commenced as drift claims, when they sometimes yielded a larger profit than they afterwards did as hydraulic claims. This kind of mining, which had almost fallen into disuse, was revived in 1872 and 1873, by the successful opening of the Bald Mountain claim of Forest City. At this place the North and Middle Yuba rivers cut three distinct channels of the Pliocene rivers at right angles, their course being nearly east and west. The bars in these rivers had early attracted attention. Their richness was due to the erosion of the sand of these ancient channels. Of these channels the more easterly one has been covered in some places to as much as 1000 ft. by volcanic lava and débris. The other channel, being on the border of the lava flow, has been very extensively hydraulicked. The centre of these channels is known as the Great Blue Lead. The course of the ancient river was nearly north and south, and had a grade of from 70 ft. to 100 ft. to the mile. Its bed has an altitude of 4350 ft., being 1500 ft. to 1600 ft. higher than the beds of the North and Middle Yuba rivers. It has been traced for twenty-five miles. This channel has been worked in a number of localities. In some places, in order to work more advantageously, the different companies have connected their claims. The workings of the Alleghany and Forest City were so connected with a tunnel 6000 ft. long. Between Rock Creek and City of Sixes there is a similar tunnel a mile long.

The skill required in this kind of mining is not so much in the method of doing the work as it is to be able to follow the ancient channels, which not only requires great ability in prospecting, but a very large amount of judgment and experience besides, for

the claims rarely give any exterior evidence of their existence or any surface indications of their direction. They are almost invariably located on some more or less well-founded theory with regard to their direction.

The probable location of the channel having been determined by explorations, which have settled the fact that the claim contains pay gravel, a point is selected where a shaft is sunk to the bedrock and drifts run in various directions from the bottom. The position of the channel having been determined in this way, a tunnel is run through the rimrock, so as to intersect the bottom of the channel in the shortest possible distance, in order to avoid a great length of tunnel, and when this has been done the mine is laid out with reference to its extent. If what is supposed to be the entrance of a channel is found the tunnel is commenced some distance below the seam of "rim-gravel," as it is called, and run on light grade until it comes into the gravel or is supposed to be beyond the rimrock, at which point a winze is sunk to the surface of the bedrock in order to reach the gravel. This winze is usually at right angles to the direction of the channel and is inclined at a considerable angle. The course, width, and grade of the channel are determined as near as can be with the least amount of work.

In order to determine the centre of the channel, the surface of the bedrock is followed until it is found to rise and fall upon two opposite sides. It does not, however, of necessity follow that because two inclines in opposite directions have been formed that the end of the pay dirt has been reached, for it sometimes happens that the channel is divided or that there are elevations and depressions in it. An examination must be made to ascertain whether it is really the rim or only an accidental elevation or depression towards the sides of the channel. The main tunnel is carried up the grade of the ancient stream and gangways are run towards each bank. The gravel is taken out in cars and washed in sluices. The main tunnel is heavily timbered, so as to be always protected; but the side drifts are temporary, and when the pay is removed are allowed to cave.

The depth of gravel that it has been found to pay to treat by this system is from 3 ft. to 8 ft. The average will not

be much over 4 ft. This includes 3 in. to 6 in. of the bedrock, which is always picked down and washed with the gravel, as the lower part, including the bedrock, is always the richest. At the Mayflower Mine, at a depth of about 200 ft. below the hydraulic ground, a false bedrock, as it is called, occurs. It is compact, gritty material, 140 ft. in thickness, and is quite horizontal. Below it was found the lower lead of blue gravel, which in the Paragon claim at Bath has been also reached. This is caused by the volcanic matter filling up an ancient stream. There occurred afterwards a period of quiescence, and subsequently the waters flowed very nearly on the course of the former bed and over what is now known as the upper lead. This was succeeded by another volcanic flow, which effectually closed the ancient river, compelling its waters to cut out a new channel, which was the origin of the present fork of the American River. The Forest Hill channel is an extremely interesting one, as the production of gold from it has been a million of dollars a mile in the aggregate. As a considerable quantity of ground had been exposed by denudation, the hydraulic process was adopted and successfully worked, but as the washing progressed into the hill, the superincumbent volcanic matter became thicker and tunnelling had to be resorted to. The mine was started in an old hydraulic pit which had been worked out. The shaft was sunk 200 ft. lower, and from its bottom drifts were run, and from its end another shaft was sunk. Much of this gravel is cement. The water and gravel have to be lifted 300 ft. vertically. The channel is about 4000 ft. wide, but only 90 ft. of the richest part of it, which yields from $15 to $50 a ton, is mined. Directly above what is at present being mined is a rich deposit which it is expected at some future time to work by the hydraulic method. The works are run by steam, and the working is so expensive that a tunnel is projected to run it with water.

The Deep Lead placer mine in the Mokelumne Hill district, Calaveras County, was located in 1880. During that year an incline 380 ft. long was sunk to the channel, so that its lowest extremity was 130 ft. vertically from the surface. The workings in 1885 were 425 ft. vertically from the surface, 2600 ft. having been run along the channel since the mine was opened. The

width of the channel is from 40 ft. to 60 ft. It is from one to one and a half miles long, and runs very nearly north and south. The mine and mill are run by three hurdy-gurdy wheels, with water at a pressure of 225 ft. Forty-two miners' inches are used in twenty-four hours, of which twelve are for the hoisting, twelve for the pump, and twenty for the mill.

The banks and bars of the Middle Fork of the American were noted for their richness in the early days* of river and placer mining from 1849 to 1863. The product of the river claims would now be considered fabulous. Those who were not so fortunate as to acquire claims on the river followed up the gulches and ravines, working them by "rocker," "long tom," and "sluices," until the source of the river gold was discovered, 2000 ft. above the river bed, at the line of contact between the bedrock slate and the overlying capping of volcanic material. The original locations were made in squares; but subsequently the mining laws were amended and each locator was entitled to a certain frontage, with side lines extended back to the apex or the water-shed line of the ridge. The frontage, as regulated by the mining laws of Forest Hill district, was only 75 ft., the locations extending back to the centre of the ridge. These claims were opened by tunnels, some of which attained a length of half a mile or more. The ground worked out yielded at the rate of $1,000,000 per mile of channel. The highest yield was in 1866, when the product was $440,000 from the Forest Hill group alone. The yield sometimes ran up to $10 per carload, and equalled from $300 to $500 per linear foot of channels.

The Forest Hill Blue Lead of Placer County is evidently the remnant of an ancient river, now covered with many hundred feet of volcanic matter. It may be divided, for purposes of description, into two geographical sections, the lower and upper. The lower section presented itself on the sides of the ridge at Michigan Bluff, Bath, and Forest Hill, and terminated at Peckham Hill, where it was washed away when the great cañon of the Middle Fork of the American River was eroded. Its length between Michigan Bluff and Peckham Hill, allowing for the sinuosities of the channel, may be estimated at from eighteen to

* Prod. Gold and Silver in the U.S., 1883, p. 719.

twenty miles, the greater portion of which has been partially worked out. It is probable that the side exposure at Bath and Forest Hill was caused by the breaking off of a bend or elbow of the ancient river, which had its source higher on the divide. The term "divide," in general use in the mountain regions of California, is synonymous with "ridge," and is used to designate the high land between two converging rivers. The source and termination of this channel or system of channels is purely a matter of theory. Its rims of slate rock are well exposed, and are sometimes more than a mile apart, and its current must have been more than a thousand feet wide at stages of high water. The ancient channels of this region consist of detached fragments of the original deposit, cut off in continuity by the deep cañons of the present rivers. The channel has been several times lost but it has always been re-discovered.

The stratification of this channel was well exposed by the air shaft of the Paragon claim, between Bath and Forest Hill, as follows:

	ft.
Volcanic matter	280
Quartz gravel	12
Compact quartz sand	30
Sand and gravel (soft ground)...	70
Pay gravel of the "upper lead"	8 to 10
Total	402

At this point the upper strata were drifted to a width of 400 ft. The upper stratum or "lead" was worked for more than twenty-five years. Below this occurred poor strata for a depth of 150 ft., when the lower stratum or bedrock was reached, which proved richer than the upper. The gravel, like most of that in this district, is so hard that the bottom stratum has to be blasted and crushed or exposed to the weather for a long time to slack—a method which at one time was very general—before it can be treated in the sluices. At no time since the discovery have operations been totally suspended on this claim, though there were periods when the channel was temporarily lost.

The Hidden Treasure Drift Mine, at Sunny South, Placer County, is located 500 ft. from the top of a very steep cañon. It is a very large and important one, having the same white character

of quartz as the Mountain Gate of Damascus, and is situated four miles west of it. Though not yielding much over one dollar per carload of gravel, it is so extensive and easily worked as to make it one of the most profitable mines in this county. Work on this claim was commenced in September, 1876. Six years afterwards an upper tunnel was driven on the channel varying from 2 ft. to 600 ft. in width, and which averaged 300 ft. This channel runs nearly north and south, its grade being to the south about 70 ft. to the mile. The tunnel, which is 4000 ft. long, is run at this grade and keeps 30 ft. below the surface of the bedrock and in the channel. It is 7 ft. 4 in. high, 8 ft. on the bottom, and 3 ft. 6 in. on the top. It is tracked the entire length with T railroad iron. From sixty to seventy men are employed, and 400 carloads of gravel extracted from it every twenty-four hours. The outside plant is an excellent one in every respect. It is worked by a company of miners who are its owners. It produced in 1883 $104,965, and in 1887 $91,000.

The Mountain Gate is located at Damascus on the northern end of the same channel as the Hidden Treasure, and is only a few miles from it. It is also on the side of a very steep cañon. It was formed by the union of the Golden Gate and Mountain Gate claims. The claims were located in 1852, and embrace 1004 acres of ground. The channel has an average width of 220 ft. It is worked, like the Hidden Treasure, by an association of miners, who own the stock, and a majority of whom work in the mines and furnish a substitute when they are absent. They have worked about 20,000 linear feet of channel and drifts. Three tunnels have been driven, the one below the other, necessitated by there being two channels and the fact that the bedrock pitches, so as to carry the pay dirt below the level of the first tunnel. The first channel was lost by being cut off by a lava flow. The channel has the same quality of gravel, the same grade, and is undoubtedly the same as the Hidden Treasure channel. The main tunnel is 9000 ft. long, 6 ft. wide, and 7½ ft. high; 6500 ft. in the channel is sharply cut off by lava or cement, as it is called by the miners. An incline in this cement discovered another channel running north-east and south-west 80 ft. lower than the original channel. It is about the same width

as the other, but contains an entirely different kind of gravel, consisting of granite pebbles and boulders instead of quartz. The character of the gold is also different, being flake gold, nearly uniform, and of slightly finer quality. It contains a larger and richer body of gravel than the other channel, but it is worked at a good deal of disadvantage. Water and gravel have to be raised from this lower channel, now being worked. For this purpose the mine waters of the upper workings have been utilised by an overshot water-wheel 40 ft. in diameter and 6500 ft. from the entrance to the tunnel. Great trouble is found in securing the practical working of the machinery. Recently there has not been power enough to unwater the mine. The gravel in the upper channel is white quartz, moderately rich in gold, but in such large quantities and so easily taken out as to admit of its being profitably worked. The lower channel is blue in colour and of granite and slate origin. The gold is very uniform in size and fineness. It is worth $18.60 the ounce. In 1883 twenty men were at work, all being owners in the claim. For several years the mine has not paid; its largest yield in one year was $63,000. The tunnel of the Bob Lewis claim east of the Golden Gate and on another channel is 1200 ft. long. At the Bald Mountain Extension in Sierra County, in 1877, the pay was entirely cut off by a dyke of basalt, which it was necessary to go round, when the pay was again found. In 1884, an uprise struck a rich channel. An incline in the volcanic rock 240 ft. was run up to the tunnel, which is 6000 ft. long. On this incline the loaded cars bring up the empty ones.

A drift mine is always worked through a tunnel run into it from the hillside or from a ravine whenever practicable, so that its bottom will be either on or a little below the bedrock with only just enough inclination to insure drainage. They are only worked from shafts when it is not practicable to reach the pay gravel in any other way. In the early days of drift mining the necessity of driving long tunnels was often the presage of complete failure. All the work of driving having to be done by hand, the great length of time required to drive them and the cost of the enterprise often either prevented the undertaking altogether or caused its abandonment from discouragement

after six, eight, or even ten years of work on it, the companies disbanding in sheer weariness. In these days of power-drills such enterprises are not feared, and are willingly undertaken. The location and construction of this tunnel is of the greatest importance. The success of the mine depends upon it. All the ground should have been previously prospected in order that this tunnel may be correctly run. Whenever it is impossible to reach the gravel by a tunnel, or where the latter would be very long or expensive, it is necessary to use vertical or inclined shafts. The first cost of reaching the gravel by a shaft is less than running a tunnel, but for taking out the gravel the tunnel is desirable, for in working by a shaft it is necessary to raise all the gravel and mine water to the surface. Hoisting and pumping machinery is, therefore, necessary, and makes an expensive plant. Practically it costs about as much to bring the gravel, in any mine extensively worked, from the point where it is mined to the foot of the shaft, and load it into the cage or bucket, as to take it out in cars through a tunnel from 1500 ft. to 3000 ft. in length. A mine which is worked by a shaft must, therefore, have much richer gravel than if worked through a tunnel. Exactly how much richer depends upon how extensively the mine is being exploited and the cost of hoisting. Shafts, when used, are sunk to the bedrock usually for exploring purposes, but are not generally used for exploitation. When water is plenty the hoisting is done by hurdy-gurdy wheels.

The location, elevation, and course of the channel having been ascertained by shafts, a tunnel 7 ft. by 9 ft. is driven from the nearest practical point of the lower or down stream end of the ground. This tunnel must be securely and heavily timbered; after reaching the channel it is continued up stream, as nearly as possible in the centre of the channel, keeping either entirely in the bedrock, entirely in the gravel, or partly in bedrock and partly in gravel. The gravel overhead is supported by timbering until the pay gravel for a considerable area is removed. In order to prevent its crushing in the main tunnel the timbering is made very strong, and pillars are left in the ground 30 ft. or more on either side of the main tunnel. When the limit of the claim has been reached these pillars are robbed. It will generally be

necessary, in order to obtain sufficient dumping ground, to run an incline on the surface, from the mouth of the tunnel to the place where the dirt is treated. The inclination of this incline is usually 8 ft. in 100. If the material used is to be stamped, it must deliver it into bins. If the dirt is soft it ends in a chute, which allows it to slide into the space where it is to be washed.

As for prospecting purposes tunnelling is quite expensive, where the top of the cement or gravel is not thought to exceed 150 ft. a vertical prospect shaft is frequently sunk. Such a shaft can be made circular, and only large enough for one man to work at a time. It requires little or no timber, needs no hoisting machinery except a windlass, and costs, according to the hardness of the gravel, from $2.50 to $12 per foot. Drifts are run from the bottom of this shaft on the bedrock, and the richness of the gravel ascertained. It is always advantageous to prospect by shaft because it is so much cheaper than tunnelling. It opens up much more ground with the same expenditure in very much less time, but is uncertain because the quantity of water which may come in may be so great as to make it beyond the control of the prospector.

It is not always practicable to open up a claim systematically. Sometimes the tunnel is not run low enough, and, instead of remaining in the bedrock, comes out into the gravel. If in this case the bedrock pitches off in the direction in which the work is being prosecuted, it may be necessary to run a new tunnel lower down. Whether necessary or not will depend principally on the amount of water in the mine. If the tunnel has been run too low the gravel from the drifts will have to be dumped into a chute to be loaded into the cars below. If it is run too high an incline will have to be run to the bedrock and the gravel hoisted. When the expense of a regular tramway is not justified the gravel is brought in large buckets, which are turned on their side and hoisted on two pieces of timber laid far enough apart to catch the sides of the bucket.

When the ground has been thoroughly prospected, and the direction and width of the channel is known, the tunnel which is to serve the whole ground is run. The entrance, in order to secure, first, that the tunnel may be as short as possible, and

second, that it may have the best facilities for washing, is generally commenced in a small ravine, which usually cuts into the mountain. Its direction should be such as to bring it in the bedrock under the nearest point in the centre of the channel. Occasionally the conditions are such as will admit of its being run entirely in the gravel, but these are very exceptional. From this point the tunnel is run in the bedrock on the line of the channel as the developments in the mine show what the course is to be. To keep the tunnel in the bedrock its entrance must be much lower than the channel bed. What this depth must be depends upon the grade of the channel, its flow, the grade of the tunnel, and the length of the ground it is intended to work. All these, excepting the grade of the tunnel, are at the commencement of the work determined approximately only. It is better to have a tunnel much too low than a little too high, so a very liberal allowance is very generally made to cover possible errors of calculation from uncertain information. If the first tunnel is too high, it often necessitates running one or more tunnels lower down, or working with interior shafts or winzes. The dimensions of the main tunnel, formerly adopted, were: height, 6½ ft.; width, when not timbered, 5½ ft. at the bottom, narrowing towards the top. The larger size of 7 ft. by 9 ft. is a better dimension. The posts are made from 10 in. to 20 in. in diameter, and for special purposes a diameter of 2 ft. is not uncommon. Caps are geneally 12 in. and upwards. The original method of making a nearly square tunnel has been abandoned in nearly all the large mines, and the system is now to make the main tunnel, for a single track, 7 ft. on the bottom, 6½ ft. to 7 ft. high, and 3½ ft. at the top, these measurements being in the clear. When double tracks are used the width is made 12 ft. on the bottom. The object of placing the posts slanting is to prevent rapid crowding at the bottom in soft ground, which, with the posts set perpendicular, would cause very great delay and expense for repairs.

The tunnel is run into the mountain on a rising ground so as to secure drainage and make the transportation of the gravel easier. Grades varying from ¼ in. to ½ in. in the rod are the lowest which are used. Those more usually employed are 3 in. to 4 in. In general, the lighter the grade consistent with

proper drainage, the better. As the bottom of the channel, when reached, is never regular, and the grade of the tunnel must be, it will often cut into the bedrock, which is no disadvantage when it contains gold, as it often does. Occasionally it will be in the gravel, but these irregularities are accidental and do not affect the general working. When the rock in the tunnel has a tendency to cave, to slide, or to swell, it must be heavily timbered. If necessary, lagging is put on the top and on the sides. Timber which has been roughly squared with an axe is generally used. All that is to be used in the mine is generally cut, dressed, and delivered by contract at 8 to 12 cents for the single stick. Lagging costs usually from $20 to $40 per thousand. The wood used is generally the pine, spruce, fir, and cedar, which covers the ground of the claim, or grows near by. Usually enough can be obtained from the surface of the claim to supply all the wants of the mine. Men are employed by contract during the summer cutting timber and lagging.

The cost of running the main tunnel and sinking the shafts in the bedrock depends entirely upon the character and hardness of the rock. In slate or other soft rock it will cost as low as $6 per foot, including the timbering and lagging. Harder rocks cost as much as $15 to $20. In blasting the harder varieties of rock, giant powder is used. Black powder is found most efficient in serpentines and similar rocks. The most economical way for the mine owner to run a tunnel, in fact, to do all the underground work, is by contract. When the main tunnel is run underneath the centre of the channel, a winze or an incline, the winze being preferable, is made through the bedrock to the gravel. If it is not intended to continue the work any further, an incline is sometimes preferred, especially if the timbers and other material used in the mine are to be taken through it. They are generally constructed in two compartments, one for the gravel and the other, containing the pipe, for the drainage and the ladder for the men. At the Bald Mountain Extension Mine in Forest City the incline to reach the channel is 260 ft. in length through lava, and makes a perpendicular rise of 60 ft. It is 50 ft. above the main tunnel. This incline is, for a little more than half its length, a double track, and at the junction of the single and double tracks there is

an automatic switch, which is set by the descending loaded cars. It takes about thirty seconds to lower two loaded cars. A $\frac{3}{4}$-in. wire cable is generally used in hauling up five cars at one time. The empty ones are side-switched, and wait to be carried into the mine.

The drainage of the tunnel must be done by natural means if that is possible. Where there is a bedrock tunnel below the workings it is done through the lower tunnel. Usually there is a large amount of water in these channels. If the quantity to be handled is considerable, the water which collects in the mine workings is conducted to a main sluice when the tunnel is in the gravel, or when the tunnel is in the bedrock, to the several shafts, and, passing through them, runs either in a trough made under the tramway, or into a ditch at one side to a large reservoir tank at the entrance of the tunnel, in which it is stored to be used for washing the gravel. A pump of considerable capacity is almost invariably required in such mines when they are worked with vertical or inclined shafts. The flow of water usually increases as the mine is more completely opened, and is generally rather beyond, than within, the control of such pumping machinery as can usually be had. It sometimes happens that the tunnel running too high, will not permit of natural drainage, being from 10 ft. to 20 ft. higher than the bedrock of the channel. In this case the running of a lower tunnel can be obviated by collecting the water in a sump close to the tunnel, and emptying it by means of a syphon. In some claims hurdy-gurdy wheels are used. The power developed usually ranges from 5 to 20 horse-power for each wheel. Sometimes small overshot wheels are used for pumping and for driving air compressors, and for motive power in the cement mills, but the wheels are usually of the hurdy-gurdy type. The same power is used for hoisting. At the Mabel Mine at North Bloomfield the hoisting machinery of the dump-house was driven by a wheel 20 ft. in diameter. The water struck the wheel from a 2-in. nozzle under a pressure of 150 ft.

At the Golden Gate Mine, instead of allowing the waters to run into the lower tunnel, they are collected in a reservoir cut out of the bedrock at the inner end of the old workings, 6500 ft. from the outlet of the tunnel. From this tank there is a fall of 40 ft.

to the lower tunnel. Two overshot wheels 36 ft. in diameter are run from this tank, and furnish power sufficient to pump all the water of the lower tunnel and raise the cars of gravel. From the top of the incline the loaded cars are drawn through the tunnel by horses, and when empty are returned in the same way.

The method of exploitation is not at all unsimilar to that which is employed in ordinary coal mining. When the pay gravel is reached it is cross cut from rim to rim, gangways are run at right angles, and the ground is stoped out in breasts. The main tunnel is always run ahead of the gangways. Pillars from 20 ft. to 30 ft. wide are left on each side to support it. The tunnel should follow as nearly as practicable the lowest point or gutter of the channel. The breasts are supported by temporary posts, or by piling up the large boulders and rejected materials which have been found in the course of mining. As each breast is worked out the ground is abandoned and left to cave. The caving never takes place so suddenly as to endanger the lives of the men. Occasionally the swelling of the bedrock below hastens the closing of the worked-out chambers.

Whenever the tunnel is run too low it is connected with the gravel by a slope, or when it is run too high the ground which can be worked with the first tunnel is used, and another tunnel must then be made below, for which reason they sometimes begin too low with the expectation of working up. The main gallery is generally 10 ft. or 12 ft. on the bottom, 8 ft. or 9 ft. high, and 3 ft. or 4 ft. wide on the top. The breasts are usually 3 ft. to 5 ft. in height. At the Golden Gate Mine the gravel is only drifted out 6 ft. in height and 15 ft. in width, as this is all that will pay. When the gravel must be carried to the surface by vertical or inclined shafts, the kind of motive power to be used depends on circumstances. If there is but little gravel it may be raised by hand; if a large quantity, mule, horse or steam power is used, or water power, in the shape of a hurdy-gurdy wheel, when water can be had. When there are no shafts or inclines but little power is required, as the grade of the tunnel as at the Dardanelles, the Hidden Treasure, and other mines is usually sufficient to run out a train of two or three cars with one man at the brakes. At the Bald Mountain Mine the mine water from the upper workings

is collected in a sump excavated in the cement below and is used on an overshot wheel which raises all the water and gravel from the workings below. When the quantity of gravel is very large, mine locomotives must be used. The cars with the timber and tools in them are returned by mules except when mine locomotives are used.

At the Mabel Mine at North Bloomfield the channel is not over 100 ft. wide. The gangways are, consequently, very short. They are run from the main tunnel, 70 ft. apart, alternating on opposite sides, so that the distance from the centre of the gangway is 35 ft. A pillar of 30 ft. is left for safety on each side of this tunnel, and carefully preserved, as it constitutes the reserve of the mine, and when the deposit has been worked out is robbed. As soon as the gangways have reached 30 ft. from the main tunnel, breasts are started, and all the ground that will pay is drifted out. Gangways parallel to the main tunuel are run outside of the breasts. The ground is broken, working forward from gangway to gangway, taking out all the ground that will pay, which sometimes is as much as 25 ft. to 30 ft. high. Sometimes the largest part of the gold is found within a foot of the bedrock. In such a case only just enough gravel is removed to enable the mines to work. This will make the workings from 3 ft. to 3 ft. 6 in. high. Generally from 4 ft. to 6 ft. is rich enough to pay. The cases where it will pay to take more than this are exceptional.

At a comparatively small distance above 20 ft. to 25 ft., there may sometimes be another stratum of gravel that it will pay to drift. Three such pay streaks were discovered and worked in the Paragon claim at Bath, 1½ miles from Forest Hill, Placer County. The first was about 6 ft. on the bedrock, and some 10 ft. or 15 ft. higher a stratum of pay of about the same richness was found; then a third seam 150 ft. above the bedrock. This was worked through a separate tunnel. This mine is one of the oldest and most prosperous in the State. Its main tunnel is 1¼ miles long. It is 7 ft. high, 8 ft. wide on the bottom, and 3½ ft. on the top. The roof is supported with ordinary posts and caps until the breast is worked out, when they are removed and the ground allowed to cave. The gravel is thrown

by the drifters as near to the gangway as possible, when it is loaded into buckets or cars by the carmen, and pushed to the foot of the incline to be hoisted. When the distance to the foot of the incline is not great, the men can easily do this work. The men also run in the empty cars on their return. When the distance increases this work will be done by mules, or by a locomotive. The empty cars will be carried to the gangway, when they will be taken by the men as before. When the mine is in regular work after opening, transportation by men is too expensive. The rails used weigh 16 lb. to the yard; they are the ordinary T rails. The gauge of the track is 18 in. The cars are the ordinary box car, which holds three-quarters of a ton of gravel, or platform cars made as low as possible, so that the men can easily roll large rocks on to them. Whatever rocks are taken from the mine are, as a rule, taken out entire. Occasionally one is found to be too large. It is then blasted by laying a piece of giant powder on top of it, covering it with sand, and then exploding it. This is, however, only done in the opening of the mine. As soon as the breasts are well opened, and the bedrock exposed and well cleaned, the rocks taken from the face are piled into the empty spaces and left there.

The gravel of the Mabel Mine is quite tight and requires blasting. The holes are made 20 in. deep, loaded according to the toughness of the gravel with from one-half to two sticks of giant powder. When it is necessary to enter the bedrock, rock drills are used. The main tunnel is always run by gangs of picked men, who work the tunnel in eight-hour shifts. It is kept as far ahead as it can be, and as near the centre of the channel as possible. The gangways running off to the rims show the deviation from the centre of the channel where there is any. About four carloads of gravel to a man is broken. This includes all the men at work in the mine. It varies somewhat with the amount of gangway work, the number of large rocks found in the gravel, and the amount of bedrock to be cut up. At the Golden Gate eighty carloads of 1½ tons are taken out daily, worth $1.75 per ton. This deposit is very extensive, but it is not rich. Forty men are employed in the mine; the day's wages is $2.50. The gravel on the Bob Lewis claim is worth $6 to $8 per carload. The Dam

claim yields $4. The Hidden Treasure yields $1.50 to $2.50; 250 to 300 carloads are extracted daily, and sixty to eighty men in twelve-hour shifts employed.

The amount extracted during several months and its value at the Mabel Mine is given below:

VALUE OF GRAVEL EXTRACTED AT THE MABEL MINE.

Month.	Gravel.	Value per Ton.	REMARKS.
1887.		$	
September .	2796	2.12	
October .	3027¾	1.40	
November .	2065½	1.85	
December .	1185	4.89	General clean up.
1888.			
January .	1497	4.89	General clean up.
February .	3052		
	13,623¼		

The ventilation of the tunnels requires considerable attention, but is rarely ever difficult. The workings in the mine are sometimes connected with the surface by a tunnel or shaft, or both, independent of the bedrock of the tunnel or by an incline. In such a case there would be a temporary tunnel in the drifting ground and a permanent one in the bedrock. The tunnel then becomes an airway, and is connected with all the cross drifts on one side of the ground which is being drifted, and not through the middle of it. This gangway is always well timbered, and the ground in its vicinity must not be disturbed while the mine is being worked, as any caving which would close it might result in serious accidents and loss of life. It is used not only for ventilation, but for access to the mine and for traffic. When there is not a sufficient difference of elevation of temperature between the two tunnels to cause the air to circulate, a shaft is made connecting the air-gangways with the surface, and an artificial draught is made by the use of fire. When there is only one opening in the mine ventilation must be secured by fans, which are run usually by water power. The air is usually carried through the mine in sheet-iron pipes.

The main and cross drifts are run in the gravel on the surface of the bedrock if it is hard and contains no gold. If the bedrock is soft, and does contain gold, 8 in. or 10 in., sometimes 2 ft.

of it, are taken up. The ground is divided into squares for the purpose of prospecting. The gravel from different drifts is washed separately, in order to determine the probable value of the adjacent ground. Where there is hardly any uniformity in the distribution of the gold in the gravel this is not done to any extent. In estimating whether or not a certain amount of ground will pay, the gross yield, or the yield per carload, is not the only thing which it is necessary to take into account. The amount of timber or lagging, and the hardness of the gravel, must be taken into consideration. No general rule can be given as to what gravel will pay and what will not. The degree of richness which makes mining profitable must be determined in each special case. In taking out the gravel only the finer portions are taken out and washed. All the large boulders are piled up in the empty chambers which have been worked out. As soon as a gangway has been worked in 7 ft. or 8 ft. the track is advanced to the heading, so as to bring the cars close up to the face. When this is again worked off 7 ft. or 8 ft. in depth the track is again moved and the cars brought up again. When 10 ft. or more of gravel is mined the breasting is done in two benches. Very long timbers are used, and if the gravel is soft and liable to cave great care must be taken to place them and the lagging properly. Except where the gravel is very hard and cemented, large amounts of timber are required. In many mines, in addition to timbers and caps placed closely together, it is necessary to use top lagging, driving the latter ahead of the timbers, and close up to the face as the gravel is taken away. Blasting is only required in extremely hard and cemented gravels. In most of the mines the ground is so soft that it can be easily worked with a pick. As far as practicable the gravel is broken with the pick, as it is difficult to use powder to advantage. On the Blue Lead in Sierra County the gravel is quite soft. It is washed by projecting a stream of water under a great head of pressure upon the loosened dirt.

The cars and track used in the gravel mines, when they discharge through shafts to a lower rock tunnel, are somewhat different in construction from those used for carrying the gravel to the dump. The track in the workings has to be moved

frequently, and is often made in sections so as to be taken up without difficulty. Sometimes wooden rails are used, but generally, if used, they are faced with strap-iron 2 in. wide by $\frac{1}{4}$ in., $\frac{3}{8}$ in., or $\frac{1}{2}$ in. in thickness; 1 in. thick, however, is found to be the best. Wooden rails, however, whether strapped or not, are being rapidly replaced by iron or steel T rails. Those generally used vary from 16 lb. to 42 lb. to the yard. The use of iron or steel rails has a great advantage over wooden rails, because, when the ground heaves, as it very often does, the rail rises in long swells, instead of making sharp bends, as with wood. In the main tunnel, and in all the drifts which are permanently used, the tracks should be laid with great care. The wear and tear of the rails is large, and frequent repairs are necessary. The gauge of the track varies from 16 in. to 22 in. The size of the mine car is generally from $4\frac{1}{2}$ ft. to 5 ft. long, 24 in. wide, and 20 in. to 30 in. deep. They are generally calculated to hold about a cubic yard of loose dirt, or from 600 lb. to 700 lb. to a ton. In making these calculations it is computed that 17 cubic feet of clay, 18 cubic feet of earth or gravel in the bank, 25 cubic feet of sand, and 27 cubic feet of broken earth or gravel when dry, will weigh 1 ton. The cars are generally made of oak, but iron cars are sometimes used, and are much better, both in durability and economy of running, and besides they weigh less. Both box and platform cars are used. The gravel is shovelled into, or on to, these cars, which in the mine are moved by hand, mule, or horse power. When a shaft is used it is loaded into cars or buckets on wheels. The mine car, which carries the gravel from the breast to the chute, is only half the size of that which carries the gravel from the chute to the outside. These cars carry from 400 lb. to 1000 lb.

The mine cars are dumped into a vertical chute with a gate at the bottom, which holds the gravel until an empty dump-car is ready to be filled with it. These are generally run out of the mine by gravity, and carried back by men or animals. It is best to use horse or mule power rather than hand. At the Hidden Treasure there are three trains of twenty-six cars each, one loading one on the way and one dumping. They come out by gravity and are returned by horses; the cars carry 1 ton.

Twelve trips are made in ten hours. The gravel averages $1.40 to $1.50 the carload. Occasionally, when the tunnels are very long, mine locomotives are used, as at the Bald Mountain Mine, where sixteen to twenty cars are drawn in, and fourteen to eighteen brought out in a trip. This cannot be done, however, unless the amount of material to be moved is very great. It was found at the Bald Mountain Mine that tramming by men cost 21 cents the load, by mules 9 cents, and by locomotive $4\frac{3}{4}$ cents per load.

The Derbec Mine, three miles from North Bloomfield, Nevada County, was opened in the year 1877. Its length of channel, including the Mabel Mine, is two miles, and it is estimated that there are six more to be worked. The greatest width of the channel has been 700 ft., the least 175 ft.; it has averaged 450 ft. The claim is two miles long, but the channel bends around in it, so as to make the total length much greater than that. The elevation of the mine is 3878 ft. The greatest depth from the surface is 536 ft., the lowest 466 ft.; it is worked by a shaft 466 ft. deep. The mine is ventilated by a Root's blower, which enters the tunnel connecting with the shaft, so that in case of fire on the surface the men have a chance for escape. The length of the main drift is 4800 ft. The extraction is done in the mine by mules, who wear candles placed in a lantern in front of their breast. The cars are of iron, 4 ft. by 2 ft., and 2 ft. deep; they hold 1500 lb.; they are raised vertically by steam. The average yield of ground is $2.50; it has sometimes yielded as high as $9; the lowest that has been found has been 70 cents. The least that it will pay to work is 90 cents. The shaft is very wet, and the mine is correspondingly so. All the water goes out through the Mabel tunnel. The pay seam on the bedrock is usually 12 ft. high. When it leaves the bedrock, as it sometimes does, it is usually 8 ft. above it and becomes about 8 ft. thick. The rocks and boulders found in some parts of the mine are completely decomposed, in others they are still in their natural condition. The gravel is generally soft enough to yield to the pick, but sometimes has to be blasted. About one-third of a pound of powder to the ton extracted is used for the purpose. In the years 1887 and 1888, 14,350 lb. of powder were used to break

60,425 tons of dirt. The powder used is giant powder, No. 2, star. The value of the ground is estimated by the ton, which is about one and one-third wagons; 300 wagons a day are hauled from the mine; the shifts are ten hours. The mine is worked by forty-two men, eleven of whom are on the surface; this includes all the men in the dump-house. The gold found on the bottom of the bedrock is generally coarse and of less value than that on the rim, which is finer. The average value of the ounce is $18.50. The miners' inch is the same as at Bloomfield. The gravel from the mine is raised vertically from the shaft and dumped into a dump-house, the wheels of the car being caught, so that the whole car is tipped and the gravel falls on to the floor below.

The Mabel Mine, near North Bloomfield, is a continuation of the Derbec Channel. It was worked from the lower side into that channel as well as worked from the surface, with an incline 225 ft. long, and was drained by a tunnel which now drains the Derbec Mine. It was opened in March, 1885, and worked to February, 1888, when it was worked out. The length of the claim was 2000 ft.; the width of the channel averaged 100 ft.; the least height of the ground to the surface over the channel was 25 ft., the greatest was 200 ft. The total length of tunnel in the mine was 2300 ft. From the bottom of the incline to the dump-house was 925 ft. The power used for extraction was an ordinary hurdy-gurdy wheel 20 ft. in diameter. The cars were of iron and contained 15 cubic feet. The value of dirt was estimated by the ton; the least that paid was $1.25. The tails were washed but once; a second washing did not pay day's wages. The miners' inch used was the North Bloomfield, which is 2230 cubic feet in twenty-four hours. They used 200 in. of water under a head of 126 ft. The dump and prospect house was similar to the Derbec. Mercury was put in at the head of the sluice, but most of the gold was found within the first 20 ft. The amount of mercury used was 76½ lb. per month. The yield of the clean-ups was $6800, $7400, $4350, $5000, $6950. The gold was .898 fine. The clean-up was made once a month. The rimrock was only 10 ft. to 12 ft. above the bottom of the bedrock. For the clean-up an arrastra of iron, 4 ft. in diameter and 22 in. high, with two drags of cast iron weighing 200 lb., was used.

The cost of the carload of gravel at the Mabel Mine, in September, 1886, at the foot of the incline, was, including every item, $1.12. The cost of hauling to the dump-house, $0.07 per load, or a total of $1.19 per load delivered at the dump-house. This cost is itemised as follows:

	$
Timber, per carload	0.02
Powder, fuzes, caps, and crushers	0.09
Labour, superintendence, &c.	0.88
Royalty and rental...	0.20
	$1.19

The yield was $1.55, leaving a net profit of $0.36 per carload.

The method of stating the yield varies. In some places the standard is the carload, which is calculated at 16 to 20 cubic feet for one ton; but, as the size of the car depends upon the size of the tunnel, the carload, without giving the dimensions, cannot be any standard of value. It is supposed that 27 cubic feet in the car represents 20 cubic feet of material in place. The expansion of the gravel, after it has been broken, is a matter of considerable consequence. Nine cubic feet of ground in place, when broken, will fill a box of 12 cubic feet capacity. An estimate, based upon cubing the feet without explanation, would, therefore, be entirely erroneous. This distinction between ground in the mine and ground broken should therefore always be maintained. In the mine it is so compact as often to require to be blasted. Experience at Bald Mountain has shown that one cubic yard of gravel in place will fill two cars. Another way of estimating the yield is per foot of channel. In some places the value is expressed as so much per acre, or as so much per superficial yard of channel ground, but the height of the ground is not uniform. In some places it is 3 ft., and in others 4 ft. or 5 ft. Taking the acre as a standard it is estimated that the yield throughout the State has been from $20,000 to $80,000 per acre, or not less than from $200 to $300 per linear foot. The gravel worked in the Forest Hill district previous to 1860 yielded nearly $250,000 per acre. The only way to definitely express the value is to express it by the cubic yard, but this method of expression is not in general use, so that it is not easy to get a standard of value for comparison between different mines.

The yield has formed the subject of many investigations during the last few years, as the opinion prevails that the future product of the gold in California for the next twenty years will depend largely upon this source of supply. The gold yield of these placers is very various. Single carloads of gravel have been taken out that paid $11,000. $800,000 was at once taken from half an acre of ground; but generally the gravel is not so rich. Mines that have run upon old river bars, such as were always found in the ancient river channels, are very common; as in every other enterprise, only average ground can be counted on.

The product and rate per foot of channel of the leading mines between 1870 and 1880* is given in the Table below.

Name.	Length of Location.	Yield per Cubic Foot.	Total Yield.
		$	$
Union . .	2400	625.	1,500,000
Hawkeye . .	800	437.50	350,000
Pittsburgh .	860	506.	420,000
Monumental .	1040	312.	325,000
Empire . .	1560	482.05	752,000

The following Table† gives the value of free gold of various fineness per ton (2000 lb.) of quartz or cubic yard of gravel:

Weight Washed Gold. 4-lb. Sample.	Fineness, 780. Value per Ounce, $16.12.	Fineness, 830. Value per Ounce, $17.15.	Fineness, 875. Value per Ounce, $18.08.	Fineness, 920. Value per Ounce, $19.01
grains.	$	$	$	$
5	83.97	89.36	94.20	99.05
4	67.18	71.49	75.36	79.24
3	50.38	53.61	56.52	59.43
2	33.59	35.74	37.68	39.62
1	16.79	17.87	18.84	19.81
.9	15.11	16.08	16.95	17.82
.8	13.43	14.29	15.07	15.84
.7	11.75	12.51	13.19	13.86
.6	10.07	10.73	11.30	11.88
.5	8.40	8.93	9.42	9.90
.4	6.71	7.14	7.53	7.92
.3	5.03	5.36	5.65	5.94
.2	3.36	3.57	3.76	3.96
.1	1.68	1.78	1.88	1.98

* Prepared by Mr. E. S. Thurston.

† Prepared by Melville Atwood, of San Francisco.

To determine the gold value of a cubic yard of auriferous gravel, a sample of 60 lb. of gravel is pulverised, and carefully washed by batea or pan; the gold amalgamated; the quicksilver volatilised; and the button weighed. In the column in the foregoing Table, opposite the weight, will be found the gold value of a cubic yard of the gravel. If a sample of 60 lb. produces a button weighing one grain, the fineness of the gold being 780, the value of one cubic yard of such gravel would be $1.67. This is arrived at by dividing the value given in the Table by 10. If the sample of 60 lb. yields a button weighing one grain and two-tenths ($1\frac{2}{10}$ grains), then the value of the gravel per cubic yard would be—gold being 920 fine—as follows:

1 grain...	... 920 fine...	... value $1.98
$\frac{2}{10}$,, ...	... 920 ,, ...	... ,, .03+
Total value cubic yard		... ... $2.01+

This Table is prepared upon the supposition that a sample of 4 lb. of quartz is the one five-hundredth part in weight of a ton of 2000 lb., and the gold values given are reduced to this proportion. Eighteen cubic feet of gravel in bank will weigh 1 ton, or 2000 lb., and a cubic yard, or 27 cubic feet, will weigh 3000 lb. or $1\frac{1}{2}$ tons; and 60 lb. being the one-fiftieth part of the weight of a cubic yard, then the relative proportion of the weight of quartz to gravel is as 50 to 500, or 1 to 10.

In order to make it pay, the gravel should contain, under ordinary circumstances, at least, from 50 cents to $1 per carload of 16 cubic feet. The minimum value of ground to be worked should thus be from 50 cents to $1 per cubic yard of gravel extracted.

The gold is extracted either by sluicing or milling. The sluicing is in every way similar to that employed in ordinary sluice mining. The milling process is adopted where the gravel is so cemented together that it cannot be sluiced. It is then crushed in an ordinary stamp mill. The way the gravel is to be treated to extract the gold depends upon its texture. When the material is compact but not cemented, it is readily washed. The gravel is dumped into a bin, which has a riffled sluice at the bottom. A stream of water is turned on to this, and the gravel is run through several hundred feet of riffled sluices charged with quicksilver. The pipe which is used to wash the gravel is generally a

2-in. to 4-in. nozzle, and the water is projected under a head of 50 ft. to 100 ft. or more The sluices are usually from 18 in. to 20 in. deep, and are set on a grade of 4 in. to 6 in. to the rod, sometimes a little less. The gravel from some of the Blue Lead channel mines is very compact, and consequently it has to be crushed in a stamp mill before the gold can be obtained, which makes it expensive to mine, and very costly to get the gold out. The amount of water used in drift mining on loose gravel is not so large as in hydraulic mining, since the quantity of material to be treated is much smaller, but it is relatively about the same. The ordinary duty of a twenty-four hour miners' inch, equivalent to 2234 cubic feet, varies from 3 to 20 cubic yards, depending upon the character of the gravel which is treated. When gravel is cemented, and sometimes when it is loose but not loose enough to be treated in the sluice, it must be crushed in stamp mills of the same kind as those used for quartz-milling, but fitted with coarser screens. The method of treatment is always battery amalgamation with riffled sluices.

The material which has passed the sluice is generally deposited in a side cañon, and is often washed over again at some future time, in another set of sluices. These tails swell and break up further after exposure, and are sometimes washed even a third time. The tails are generally washed over after the channel is worked out. They are expected to contain about 1 per cent. of the amount obtained at the first washing, but they often contain more. Their richness depends on the amount of gravel washed at any one time, and on the care exercised in doing it. The method, however, as compared with hydraulic mining is expensive. The mine sometimes produces water enough for the treatment.

When water is scarce the gravel extracted is piled in some convenient position waiting for water supply. The Bald Mountain Company store in this way, in the dry season, from 40,000 to 50,000 tons. When water is plenty it is washed at once in the dump-house. As the gold on the bedrock is oftener coarse than fine, it is generally sluiced in the dump-house without quicksilver, but mercury is generally used in the lower sluices. As the grade of the sluice can be controlled at will, it need not be very long. At the Mabel Mine the gold on the bottom of the

channel is coarse, but grows fine as the rims are reached. The bars made at this mine from it were .8986 fine. The smaller the gold the more difficult it is to save it.

The dump for the gravel outside of the mine, and the arrangement for washing, vary according to the different positions. If the tunnel enters the mine from the head of some steep ravine, and the mine uses its own water, the tank for the mine waters will be directly in front of it. This tank is generally large enough to hold all the mine water for the twenty-four hours. Below it and running down the bed of the ravine, sluice-boxes are fixed. The track which leaves the tunnel divides, in all the large mines, into two or three branches. The main one crosses the water tank and is continued on a trestle over the sluices in the dump-house, the floor of which is made of double planking set at an angle, so that the gravel slides into a sluice in the middle of the house during the washing. A second branch runs on one side of the main dump, and is used for taking to a special dump, called the "prospect dump," the gravel from the drifts and ground it is desired to prospect, so that they can be washed separately. In small mines only one track is used for all these purposes. These tracks and the entrances to the tunnels are generally roofed over as a protection from snow and rain. The prospect dump where gravel is washed in small lots to determine whether it is rich enough to work, is next to the main dump-house. The washing is done once a day, or more frequently if the quantity of gravel is large. The sluices are cleaned up at once as a precaution against robbery. When the gold is, as a rule, coarse and heavy, it often requires no quicksilver. Some is generally placed in the lower sluices to collect the fine gold. As a general thing, the tails, after being discharged in the rivers, are retained there by means of dams to be treated again.

The dump-house of the Hidden Treasure Mine is 50 ft. long. The car-house, 50 ft. above, is 90 ft. long. The platform below is inclined at 10 deg. As the cañon is very narrow, the prospect and dump-houses are side by side at the bottom of the cañon. To separate what is to go to the main and what to the "prospect dump," there is below the floor of the car-house a hanging shield, 10 ft. long and 8 ft. wide, moved by a windlass, which

is inclined so as to throw the gravel into the "prospect house" when trials are to be made. The sluice in the centre of the "dump-house" is 18 in. wide and 18 in. deep, and has a grade of 18 in. to the box. The main sluice outside of the dump-house is 500 ft. long. The upper part of it has several switches, so that any section of it can be cleaned up without stopping the rest. This sluice is continued 400 ft. with a grade of 20 in. to the box. The first 50 ft. of the main sluice in the dump-house is paved with blocks, 18 in. square and 7 in. thick, so as to have but one block in the width of the sluice. This is followed by 20 ft. of Hungarian riffles, cast together in pieces 18 in. long with eight riffles in a set, and this by a 100 ft. of iron-strapped longitudinal riffles, alternating with carwheels, and this is followed by 300 ft. of blocks. The rest of this sluice is paved with quartz cobbles. The blocks last four or five months; the Hungarian riffles fourteen months; the longitudinal riffles fourteen months; and the wheels eighteen months. The rock pavement lasts from six weeks to two months. When the rocks are removed for a clean-up some few of them can be used again. Mercury is put in only below the Hungarian riffles. Above this, in the dump-house, the free gold is caught. Only 150 lb. of mercury a year is used. The first six or eight boxes in the dump-house are cleaned every day; the riffles and wheels every ten days; the rocks only when they are worn out. At the end of 900 ft. undercurrents with longitudinal riffles are used. They are 6 ft. wide, 36 ft. long, and movable, and can be set at any angle from 14 deg. to 20 deg. by set screws at the end. At the end of this system of sluices there is a drop of 20 ft. to a sluice paved with stone 108 ft. long; then another drop and 100 ft. more of sluice. The tails from this last sluice discharge into the cañon. The last set of sluices is cleaned only once in two months.

The dump-house of the Mountain Gate is 48 ft. long and 16 ft. wide. The gravel falls 25 ft. The prospect house is at one side. The car-house above is arranged so that the cars easily dump either into one or the other. The dump-house sluice is paved with blocks, at the end of which are two boxes of wheel riffles, then two boxes of longitudinal riffles, then 10 ft. of Hungarian riffles cast together. There are eighteen boxes of slat riffles and

three of wheel riffles, followed by eighteen of slat riffles. The upper sluice has fifty-one boxes. At the end of this there is a drop; then five boxes and another drop; then eleven boxes; then a drop which discharges into three boxes; then a drop, and a single box which discharges into the cañon. The pay runs from $2 to $6.

The dump-house at the Derbec Mine is 100 ft. long and 30 ft. high, and 33 ft. 10 in. wide. The floor has a grade of 4 per cent. It inclines at the back for 16 ft. at 10 deg. At the foot of this incline, on to which the gravel is dumped, a sluice 22 in. wide and 84 ft. long, with longitudinal riffles strapped with iron, is placed. This sluice has also a side lining. From the commencement of incline on the sides of the dump-house, which are 16 ft. wide, are sloped 10 deg. on both sides towards the sluice. The gravel is washed into the sluices with a 4-in. monitor with a pressure of 100 ft.; 300 in. of water wash 600 tons, working four times a week. The monitor is placed in the centre of the lower end of the dump-house. It is played against the sides of the pile at the further end of the house, working it up and down and on the sides, sometimes making it strike the floor to rebound on to the pile of gravel, and washing in such a way as to keep the sluice clear of large stones. As the stream does not throw to the further end of the house, the water is directed on the side to the foot of the incline and rushes up it and falls against the pile at the back. The whole object of this movement is to make the large stones roll out of the sluice instead of choking it. The floor of the dump-house is made of 2-in. planks, and over these breaking joints, 1½-in. planks are nailed. When these warp and rise the gold gets underneath them. I saw 1½ oz. taken from a distance of 4 ft. up the floor, from under this lining when five boards were removed. Whenever any of the flooring, either of the main flooring or the lining, is taken up and cannot be used again, the boards are always burned and considerable gold recovered from the ashes. If they are to be used again, they are dried and pounded to make them deliver the gold and then used. Considerable gold is recovered from the dump-house boards in this way. It always catches in the rough places so as sometimes to make them look yellowish. Most of the gold in the ordinary workings is caught in the first 50 ft. of the sluice. No mercury

is put in in this distance to catch it. After the first 50 ft. mercury is used in the sluices. They use two flasks of it in fifteen days. Inside of the dump-house longitudinal riffles strapped with iron are used. Outside of it old carwheels, which have first been heated in a hot fire to burn off the oil, are used. They make excellent riffles. They are placed with the rims lapping down stream with a cross riffle every 30 ft. to 40 ft. The wheels themselves make much better riffles than the cast iron imitations of them made for the purpose, but with a square rim, to be used when there are not sufficient old wheels to fill the sluices. The carwheels can be turned in any direction and can be used until they are worn out. The sluice outside of the dump-house is 600 ft. long with a grade of 10 in. in 12 ft. More than three-quarters of it has wheel riffles. At the end of the wheel riffles there is 18 ft. of longitudinal riffles. The rest of the sluice is paved with blocks. The clean-up varies from $5000 to $7000 and is made twice a month; 90 per cent. of this value is caught in the dump-house and 10 per cent. in the sluices. The gold varies from .898 to .908 fine. 25 ft. from the end of the sluices is an undercurrent. This widens out from the grating in the sluice to 12 ft. This is paved with rocks and set at a grade of 10 in. in 12 ft. The lower part is 25 ft. by 12 ft. and has transverse riffles 1 in. apart except for the last 6 ft. All the tails are washed over twice. It is expected to get from them one-tenth of the original yield. They are collected in a series of brush dams. Under these is a bedrock flume 4000 ft. long. The tails have already been washed twice, and the third time they expect to tail into the upper end of the Malakoff diggings.

When a clean-up is to be made the gravel in the dump-house is washed out, and a screen put up to prevent any material from the cars falling into the sluice. The side boards of the sluice are removed and the longitudinal riffles taken up. If any of the floor boards of the house are warped they are lifted and the gold taken out from under them. Every board and every riffle, and also the side lining of the sluice, are taken up and washed in clear water, which is carried by pipes to the head of the sluice for that purpose. The work of the sluice is now arranged, commencing below, four men taking out the riffles and one man

with a pick loosening the gravel, taking care not to strike the pick into the bottom boards. One man with a sluice fork, which is a fork with twelve tines 1 in. apart, and one in front of him with a rake turns over the gravel. One man with a spade shovels it up towards the head of the sluice, and one with a hoe draws it up, spreading it out as much as possible. The top gravel of the upper part of the sluice soon begins to appear free from small stones, and when it commences to show colour, it with the fine sand is collected in iron buckets. As soon as about 15 ft. have been washed down, a riffle 4 in. wide is put in 10 ft. from the head of the sluice, and behind this all the gold and sand collected in the buckets is placed, pouring it through a pan with $\frac{1}{8}$ in. holes in the bottom. This pan catches all the gold nuggets, the iron nails, and stones too large to pass. When no more will pass it is turned out on the side and carefully looked over for gold and nuggets, some of which, the size of the end of the middle finger, are often found in it. All the coarse gold in the sluice is cleaned by moving it several times up and down to get it as pure as possible, but no attempt is made at this time to clean the fine gold. The sand is taken up just as soon as the colour appears, but is not cleaned until it has all been collected. Less water is now run into the sluice, and a riffle is put in at the point where the mercury has been introduced into the sluices. All the gold and sand caught upon the upper riffle is now washed. To do this it is spread over about 50 ft. of the sluice, washed up and down, and moved about so as to remove all the foreign matter. The sand and gold are moved with a hoe and the top brushed with a whisk broom and then constantly moved with the hand, moving always up towards the head of the sluice. When the gold is nearly clean, the nails and old iron are picked out of it with a magnet. All the old iron is sorted. That which can be used is put to one side. The rest is sold to the copper works. Everything, from a watch to a jack-knife, that has been lost in the mine is found at this point. The clean gold is taken up with a scoop and carried away in a pan. It is almost clean enough to melt. When the gold gets too fine, the whole of it is swept down and a man with a pail goes behind to work it down, and especially to wash the gold out of the nail holes. The gold

which is too fine to clean in the sluice is taken up to be panned and then dried and blown to get rid of the black sand. This black sand is cleaned in an arrastra. A man then goes behind with a broom and sweeps down the whole of the sluice to the lower riffle, cleaning the whole as clean as possible. The sand is then taken up to be panned. The lower riffle is now taken away. The free gold which has been cleaned in about two and a half hours from the 100 ft. of free gold sluice is now panned and melted. The riffles at the head of the sluice are put back. They are made so as to fit the sluice, with two sets of riffles, one above and one below, held down by a transverse one. The mine cars, which all this time have not been dumping, now commence to dump, and the screen, 2 ft. or 3 ft. high, temporarily put up to prevent anything from getting into the sluice, is taken away. During all this stoppage, water is made to play on the floor boards of the house on both sides, so as to prevent their shrinking. The hammer used in replacing the riffles and side lining of the sluice is a pick, cut off so that the whole head is only 8 in. long. When two sets of riffles with their side liners are in, the whole attention of the men is given to the mercury riffles below. By this time most of the wheel riffles have been taken out, and they with the largest stones have been placed upon the sides of the sluice. Every stone and every riffle removed is washed in the sluice with great care before taking it out.

To collect the mercury and amalgam the sluice is treated in sections of about 100 ft. The sand is moved in the same way as the free gold sand. As there is some mercury in it and it moves easily, it collects behind the dam of sand which is purposely made by the men who stand in the sluice by placing the two feet across it. The mercury with a good deal of sand is caught in a scoop, moving it up stream, and is collected in iron buckets. When any of it is seen in a nail hole, a splash with the hoe throws it out. The gravel is packed hard in the bottom of the sluice and has to be moved with a pick. When the buckets are full they are taken to the arrastra. All the mercury that will drain out of the sand is put into a flask. The rest of the contents of the buckets is put into the arrastra. The arrastra used is a Cox cement pan altered to an arrastra. The holes in the bottom are

plugged with wood, on which account the pan leaked badly. Beneath the pan there is a trough in which everything is caught. Before putting anything into the arrastra the contents of the pails are sieved. What passes goes to the arrastra; what does not, has the nails and iron and the sulphurets picked out of it. They are placed in a pile by themselves. The arrastra is 6 ft. 6 in. in diameter and 30 in. high, with two 200 lb. drags of cast iron. It has a lining in the bottom 6 in. high and 1 in. thick at the top and 6 in. at the bottom. This is rounded at the sides so as to suit the shape of the drags. In the centre is a cone 2 ft. high, also fitted to the shape of the drags. The drags are 2 ft. long. The arrastra makes thirty turns a minute for two hours. In it are put the mercury, sand from the sluices, all the washings of the tank in which the pan trials of ore are made, and all the black sand which comes from blowing the gold in panning. The motive power is an old-fashioned hurdy-gurdy wheel, 7 ft. in diameter. It uses 10 in. of water with a 90-ft. head. This pan is used once a month, except when the sulphurets are very rich, when it is used twice. When the grinding is finished, the arrastra is slowed down and the pulp thinned with water. In winter it is necessary to inject steam, and this is the reason why the stone drags were given up, as they were cracked by the heat. The continuous discharge from the pan is made by a pipe 2 ft. from the bottom. To empty the arrastra, a splash sluice, which is vertical, is put up, and the contents made to discharge against it. The discharge is then made from the bottom, holding an iron pail there to catch the mercury, which settles in the pail while the rest of the contents overflows. When the pulp is all out, the arrastra is washed with a hose, the drags being turned up to one side, so as to splash out any mercury from the holes. The mercury is caught in a set of Howland riffles, on which a pail is set. The water discharges from the pail and runs into a sluice about 30 ft. long. The front of this sluice has four sets of Howland riffles, then transverse riffles in gratings 6 ft. long, four sets of riffle holes, and 4 ft. of transverse riffles. Each set is taken out and washed down to the other, keeping the mercury in every one by a dam. Within 3 ft. of the end is a riffle from which all the

sand is taken up. The arrastra sluice, as well as that of the prospect dump, discharges into the main sluice. The mercury collected is washed, freed from sand, and strained and retorted, the gold is melted into bars, cleaned from sulphurets, put red-hot into sulphuric acid and water, and rubbed with fine sand to give it a good appearance, and sent to market. The gold collected at the clean-up at which I was present weighed 313½ oz. before melting, and in melting lost 15 oz. The amalgam weighed 252.20 oz. A 70 hours' run of 300 in. of water in November, 1888, washed 2841 tons of gravel.

Each mine has a prospect dump on which the value of the dirt in the mine is determined before making any large exploration upon it. It is built exactly like the dump-house, but is smaller. At the Derbec the sluice is 12 in. wide and 13 in. deep. The sides of the house are 6 ft. wide, with a grade of 10 in. in 12 ft. The house itself is 30 ft. long. The sluice tails into the main sluice, and has longitudinal riffles. To determine the value of any part of the mine, 100 carloads are tried. Smaller samples are always taken from time to time from the cars. These are panned in a tank constructed for this purpose. The residues from this tank are always treated in the arrastra.

The Dardanelles Mine at Forest Hill was opened in 1854 as a drift mine, and was subsequently extensively worked as a hydraulic claim, but is now working exclusively as a drift mine. The width of its channel is 75 ft. to 100 ft. The rise to the rim-rock on either side is very slight, "the country" on either side being almost flat. Six feet of the gravel, including that taken from the bedrock, pays to work. Occasionally 2 ft. to 3 ft. of the bedrock is blasted out. Usually 12 in., but sometimes none of it contains gold. The gravel is cemented, and requires stamping. The tunnel is 2400 ft. long. Its entrance is in the bedrock about 20 ft. below the gravel. The cement is reached by an up-rise. The method of mining is to take out everything, supporting the roof with timbers, and 20 ft. of rock is piled on each side of it for protection. Gangways are run every 30 ft. to 40 ft. The whole of the cement for a width of 40 ft. to 70 ft. is removed and the roof supported with rock. It is never allowed to fall. All the large rocks and boulders are left

in the mine. None but small ones are ever sent to the mill. The yield of the gravel is $6 to $8. The cars are of iron, and run out of the mine by gravity, attended by one man, who pushes the car back. Each car has a lantern in front of it. The car holds 2200 lb. The mill has no crusher. The storage bin holds 350 tons, but its inclination is such that the feeding must be done by hand. The mill has five stamps, with a place for five more when their erection is justified. There are no plates in the mortar, but mercury is added according to the dryness of the plate. The screens have $\frac{3}{16}$ in. round holes. The pulp falls on an amalgamated copper plate, 4½ ft. wide and 3 ft. long. The sluice is 12 ft. long. Beyond the copper plates are four riffles, ½ in. deep and 1¼ in. wide, and the width of the sluice, cut into the wood on its bottom. The sluice discharges into a Eureka rubber. From the end of the rubber it narrows in 2 ft. to 14 in. This sluice is filled with longitudinal riffles packed with small pebbles, and is three boxes long. There is then a drop of 5 ft., then three more boxes with the same kind of riffles. The battery is cleaned up every two weeks. The copper plates are dressed every morning. The rubber and sluice are cleaned once in two months. The gold found is in thick and flat pellets. There are twenty-five men in the mine and four in the mill in twenty-four hours, and besides these there is a carpenter and a blacksmith. The mill is run by a Pelton wheel, 2 ft. in diameter, with a $\frac{7}{8}$ in. nozzle and a pressure of water of 300 ft. This wheel is not large enough for ten stamps. When water is scarce the mill is run by a 36 horse-power engine, which uses two cords of dead wood a day.

The Live Oak Mine at Forest Hill is working the pebbles and boulders of the bedrock, stamping all that comes from the mine without a crusher. The tunnel is on the bedrock, and is 2800 ft. long. Only 3 ft. 6 in. of the gravel are taken out. All the large boulders are left in the mine. The mill has ten stamps of 1000 lb. each. As the rock is soft, they crush 10 tons each in twenty-four hours. The mill is run by a 5-ft. Pelton wheel with a 2½-in. nozzle. They use 80 in. of water at a pressure of 75 ft. Their screens are $\frac{1}{8}$ in. punched. In front of the battery is a sluice 5 ft. wide and 12 ft. long. On the upper end are copper

plates 2½ ft. long and the width of the sluice. Below this there are four riffle grooves, 1 in. deep and 1½ in. wide, leaving a space of 2 ft. 3 in. between each. This sluice discharges into a Eureka rubber, the discharge sluice narrowing down to the width of an ordinary one. It has longitudinal riffles. It is 20 in. wide at the top, and in 8 in. narrows to 12 in. There are 50 ft. of this sluice. From its end there is a drop of 4 ft., then 30 ft. of sluice, at the end of which the tails escape into the cañon. The battery is cleaned once in two months.

The Mayflower claim, near by, has twenty stamps. The Morning Star claim, at Iowa Hill, Placer County, is an old claim, started about 1856. The width of the channel varies from 68 ft. to 220 ft. The tunnel is 7700 ft. long, 6 ft. high in the clear, 5 ft. wide below, and 4 ft. at the top. From 6 ft. to 8 ft. of the cement is worked, which pays on the average $7 a carload. All the rocks above 6 in. in diameter which will not pay to work are left in the mine. All those below 6 in. which it will not pay at present to treat are put into a pile outside of the mine to weather, in order to work them when they become soft. It is almost impossible to free these stones from gold by a simple washing. What comes out of the mine that will pay goes to a ten-stamp mill, which is at present run by a 60 horse-power engine with wood as a fuel. A water-wheel will eventually be used. The stamps weigh 900 lb., and drop 9 in., making sixty-eight drops. The shoes and dies are cast iron, and wear ninety days. The duty of each stamp is 5 tons in twenty-four hours. The battery screens have $\frac{3}{16}$ in. punched holes. They are made in the mill. Mercury is used in the battery at the rate of 1 lb. a day, added 2 oz. or 3 oz. at a time. The apron of each battery narrows in 2 ft. to a sluice 22 in. wide and 15 ft. long. The first 8 ft. is made of rough boards. All the sluices have a grade of 11 in. to the box. It was originally intended to give 12 in., but there was not room enough in the mill to do it. This sluice terminates in one which is 6 ft. long and locked. In this sluice there are three board riffles, set like the Hungarian riffle, equally distributed over this part of the sluice. They commence 3 in. from the top and end 6 in. from the bottom. The bottom of each compartment so made corresponds to a slot which is 1½ in.

long. Between these there are four riffles with the slant up stream, which are continued to within a foot of the end of the sluice. Two sluices from each battery run into another one at right angles to it, which is 12 in. wide. From here to the end of the last sluice the riffles are of wood, made in the shape of a V, with the point up stream. This sluice is 30 ft. long. The V riffles are made to alternate with transverse ones that form a grating. Each set of V riffles contains fourteen riffles, and the transverse riffles are sixteen in a set. Each riffle is 2 in. deep. They are made in a frame, so as to be easily moved. Two of each set are in a box. This sluice runs into another one at right angles to it, which is 24 ft. long, and is arranged in the same way. This discharges into boxes with longitudinal riffles, paved with small stones, set at an angle of 30 deg., which is rarely cleaned. These tail into the cañon. The clean-up of the battery sluices is made every Saturday. The gold collected is worth 18.50 dols. the ounce.

The gravel from the Lyon Mine, El Dorado County, California, is crushed by a 20-stamp mill driven by water. The stamps weigh 850 lb. each, and drop ninety times per minute, with an 8-in. fall. Thirty miners' inches of water under a head of 130 ft. furnish the motive power. Of the total amount of gold saved, 80 per cent. is caught in the battery. Outside of the mortar the pulp flows over a silver-plated copper apron, and then through 800 ft. of sluices. But little gold is recovered from the sluices, however. The material is crushed without previous sizing. The mill of the Deep Lead Placer Mine, in Calaveras County, has eight stamps of 750 lb. The drop of the stamp is 9 in. and the number of drops eighty-six per minute. Each stamp crushes five carloads of cement in twenty-four hours through a steel wire screen of five to the inch. The mill is run by three hurdy-gurdy wheels, one of which is 4 ft. and the other two 6 ft. in diameter; 42 miners' inches of water at a pressure of 225 ft. are used in the mill. Five per cent. of the gold caught is found in the batteries and the rest on the plates and sluices. The average value of the carload is $1.

The steam stamp mill of the Paragon Company, in Placer County, California, has ten stamps of 900 lb. each, with steel shoes

and dies, dropping one hundred times per minute, with a fall of 9 in. The capacity of the stamps is 40 tons per twenty-four hours. The screens are sheet iron, with $\frac{3}{16}$-in. round holes. Three ounces of quicksilver are added in the battery twice a day. The coarser gravel and boulders are not crushed, but only the finer and softer gravel. All of the rocks about 9 in. in diameter are thrown into a pile outside the mill, and after some weeks of exposure each rock is washed in a tank by boys in order to collect the pieces of adhering gold, which are sometimes half the size of the finger nail. The gravel is dumped from the track of the tunnel to the mill platform, where large rocks are picked out and carried to the dump heap, which is sluiced by tribulers, the company receiving one-third of the gold. All the other material is fed by hand to the mortars, which have a single discharge through screens with round holes, $\frac{3}{16}$ in. in diameter. The battery sluice is 5 ft. wide and 9 ft. long. It has four riffles cut into the wood $1\frac{1}{2}$ in. deep and 1 in. wide, but no copper plates. The sluice into which it discharges runs into an Eureka rubber. From here the sluice is 100 ft. long and 14 ft. wide with longitudinal riffles. The battery is cleaned once a month, the sluices once in three weeks. It does not pay to work gravel of less than $4. The amount taken from the Eureka rubber after four months was $400. The mill is run by steam power and consumes three cords of wood per twenty-four hours. The tunnel, in the year 1880, was 1900 ft. long. The force of men employed consists of two engineers at $7.50, two feeders at $7, three helpers at $6, one foreman at $4, one blacksmith at $3.50 thirty-one miners and carmen, $93; total, $121. The greater part of the yield of this mine comes from the stamps. From 60 to 70 per cent. of all the gold saved is caught by amalgamation in the battery, from 20 to 25 per cent. on aprons, and the remainder by a concentrator and in 150 ft. of sluices. The record of a continuous run from September 19th, 1879, to April 14th, 1880, gave the following results:

	$
Carloads of gravel crushed...	4,241
Total yield	44,835.53
Average yield per carload	10.57
,, ,, ,, ton	7.05
,, cost per carload, mining and milling, including dead work	3.94

A carload at this mine contains, on an average, 28 cubic feet, and weighs about one and a half tons. The yield per ton was, therefore, $7.05. The first 800 ft. of the tunnel gave $800 the carload.

DETAILS OF STAMPS USED ON CEMENT.

	Kind of Stamp.	Number of Stamps.	Weight of Stamp.	Number of Drops.	Height of Drop.	Screens.
					in.	
Dardanelles Mill, Forest Hill	Challenge	5	850	95 to 97	7½	$\frac{3}{16}$ round
Live Oak	Hand	10	1000	95	7½	$\frac{1}{8}$ punched
Paragon	,,	10	900	100	9	$\frac{3}{16}$ punched
Morning Star	,,	10	900	68	9	$\frac{8}{16}$ holes.
Lyon	,,	20	850	90	8	
Deep Lead	,,	8	750	85 to 87	9	5 to inch
Paragon	,,	10	900	100	9	$\frac{3}{16}$ round

At the Centennial Mine the gravel is hoisted through an inclined slope 200 ft. in length and dumped into a bin, from which it passes through an automatic feeder into a revolving cylinder of boiler iron, provided in its interior with railroad rails running in the direction of the length of the cylinder. The cylinder is 20 ft. long and 3 ft. in diameter, and is run by steam. It works like a Brückner furnace. The gravel is thrown in, and in the length of the cylinder is supposed to fall about thirteen times. The result is that it is broken and cleaned. At the power end of the cylinder there is a set of grizzly bars which are about 3 ft. long and 1¼ in. apart, which allows the fine portion to pass through. This falls into the feed-box of the stamp-mill, while the large stones fall into a car to be thrown away. The machine runs thirty to thirty-five carloads of 20 cubic feet in twelve hours, and gravel paying $2 per carload pays expenses. Cement to pay for crushing must necessarily be much richer than gravel. The expense for milling is less for cement than for quartz, as a coarse-mesh screen can be used, which delivers much more rapidly; besides, the cement is considerably softer.

The sluice is cleaned in sections twice a month, and much

ftener if the ground is very rich. A general clean-up is made wice a year, when the worn blocks in the sluice are turned over nd those no longer fit to be used are replaced. With turning he blocks last ten months.

Yield of the Paragon Claim, Placer County, California, for Six Seasons, 1874 *to* 1879.

	$
Total production for the period	243,075.87
Dividends paid	119,670.14
Average yield per miners' inch, calculated on a basis of twenty-four hour inch under 6-in. pressure ...	2.54

The monthly yield of the Mountain Gate Mine has been from $10,000 to $12,000, nearly all of which was taken from the old quartz channel. The total yield of the mine up to 1881 was $1,500,000. The gold taken out is worth $18.60 the ounce. The Bob Lewis Claim yields $8000 to $10,000 yearly, and might yield more but for the scarcity of water. The Dam claim now yields $12,000 yearly. The ground worked on the Forest Hill Divide yielded at the rate of a million dollars per mile of channel. The highest yield for any one year was for 1866, when the product was $440,000 from this group of mines. The yields sometimes ran to $10 per carload, and equalled from $300 to $500 per linear foot.

The North American, near the head of Slate Creek, has an altitude of 5630 ft. above the sea. The tunnel through which t is worked is 4400 ft. in length. About 150 cubic yards of gravel are carried out of the mine daily by mules and are washed with 50 in. of water. There are fifty miners, twenty of whom are white men and the rest Chinamen.

At the Bald Mountain Drift Claim, Sierra County, the tunnel is 3600 ft. in length. The gravel is drawn out by a locomotive, drawing, as an ordinary train, from sixteen to twenty cars, carrying about two-thirds of a cubic yard each. During the year 1884 they were treating about 150 cars a day. This amounts to 100 cubic yards a day, or about 30,000 cubic yards a year. The sluice in which the gravel is worked is 17 in. wide and 8 in. high, and has a grade of 6 in. in 12 ft. It is paved with blocks, which are 16 in. square. These blocks last ten months, during which time they are turned over once. The sluice is

cleaned in sections twice a month. A general clean-up is made once a year, when the whole pavement is repaired. The mine produces about 100 oz. of gold a week They employ seventy-five white men. The Bald Mountain Extension at Forest City treats about 34,000 cubic feet of gravel and employs seventy-five white men. The Ruby, near Forest City, treats 15,000 and employs forty men. The Derbec, near North Bloomfield, has a vertical shaft 400 ft. in depth and a tunnel connecting with it; it treats 21,000 cubic yards, employs seventy-two men and uses about 100 in. of water.

The cost of opening a large drift mine and getting it ready for work is necessarily very large, and requires beside a great deal of time. The length of the main channel is very often from 2000 ft. to 3000 ft. It is usually run in hard bedrock and must be finished before the pay gravel can be reached. On its proper location depends the whole success of the enterprise; 100 ft. in length saved represents a very large saving in cost.

The ordinary conditions which determine the expenses of drift mining vary very greatly. Some deposits which are favourably situated, in which both the gravel and water run out of the main tunnel, can be worked at a very low cost. If, however, the water is difficult to manage, the ground very loose or quicksands occur, and the pay gravel is irregularly scattered over the bedrock, the expenses may be very large. In some cases gravel yielding not over a dollar per cubic yard can be worked with a profit. In other localities, $3, $4, $5, and $6 a yard will hardly pay expenses. At the Lion Mine, El Dorado County, California, where stamps are used for crushing the cement, work can be done on cement yielding $3 a yard. At the Paragon Mine in Placer County, where there is also a crushing mill, $3 yields a good profit over all expenses. At the Hidden Treasure Mine, Michigan Bluffs, it costs about $2 per ton.

The following Tables give a fair idea of the details of the cost of some of the drift mining enterprises in the United States in the early days of this method:

TUNNEL CLAIMS NEAR DOGTOWN, CALAVERAS COUNTY, CALIFORNIA.*

——	Hammerschmidt, Hensel, and Co.	Barney, Hurle, and Co.	Bully Company.	Buckeye Company.	Deitrich and Co.
Inches of water used in sluice washing	25 to 30	30	30	30	30
" " " hydraulic washing	20				
Height of fall	100 ft.				
Supply of water lasting in the year	Six to seven months	Six to eight months	6 to 8 months	6 to 8 months	6 to 8 months.
Cost of water (sluicing) per day	4 dols.	4 dols.	4 dols.	4 dols.	4 dols.
" " (hydraulic) per week	20 dols.				
Air-shafts	One of 70 ft.; one of 148 ft.	One of 105 ft.	4	4	10
Length of tunnel at present	1500 ft.	One of 200 ft.; one of 400 ft.; and one of 300 ft.	700 ft.	1000 ft.	1200 ft.
Height of drift (all pay-dirt)	6 ft.	4½ ft. to 5½ ft.	4 ft.		
When this claim commenced	In 1865	In 1862	1870	1870	1870
Area of ground drifted out and worked	9186 superficial square yards	5600 superficial square yards			
Number of cubic yards drifted out	18.332	7500 square yards	3000.		
Length of ground unworked	400 ft.	600 ft.	1000 ft.		
Average of cement-gravel extracted per day	20 to 25 carloads; about 25 square yards.	15 to 20 square yards	Over one-half.		
Quantity of refuse left in the stope	Over one-half.	Over one-half	2000 square yards.		
" " gravel-cement washed during last year and extracted	2200 square yards	Not washed	700 dols.		
Yield of cement-gravel washed	4 dols. to 5 dols. per sq. yd.	4 dols.	700 dols.		
Total yield last year	Over 8000 dols.	Not washed out			
" since commencement of this claim	70,000 dols.	30,000 dols.			
Gold, description	Small flat particles	Both coarse and fine			
Wages paid to underground drifters	2.50 dols.	2.50 dols.	2.50 dols.		
Number of men hired	Four				
" " working	Eight	4	4	4	4
Composition of deposit	Red soil, small and large, white and blue quartz, boulders, pieces of dark blue slate, quartz, and mica slate, cemented granite gravel.	Red soil, quartz, gravel, slate, granite, sand—all cemented.	Gravel, cemented	Gravel, cemented	Gravel, cemented
Bedrock	Dark blue slate, full of cubical pyrites.	Dark blue slate, veins of quartz.	Blue slate	Blue slate	Blue slate.
Direction of tunnel and course of deposit channel	E.N.E.	E.	S.E.	S.E.	S.E.

* Min. Com. Rept., 1872, p. 72.

The following Table gives the most important details of two of the claims of St. Andreas, Calaveras County, California,* which were worked in the early days:

	Garnet Company.	Wilson & Co.
Inches of water used in sluice washing	20	15
Cost of water	$10.50 per week	$1.50 per day.
Supply of water lasting	4 months	5 months.
Depth of shaft	42 ft.	33 ft.
Height of drift on an average	4½ ft.	6 ft. to 10 ft.
Area of ground drifted upon	400 ft. by 800 ft.	300 ft. by 70 ft.
Length of ground unworked on channel	1200 ft.	900 ft.
Average of cement-gravel extracted per day	2500 lb. per man	2500 lb. per man.
Quantity of refuse (boulders) left in the drift	One-half	One-half.
Quantity washed in sluices per day (ten hours)	... ...	3 to 5 cubic yards.
Quantity of gold-bearing cement-gravel crushed in twenty-four hours' run of five stamps of 500 lb.	18 tons, or about 7 cubic yards.	
Crushed with heavy Chile arrastra	25 tons, or 10 cubic yards.	
Expense of milling, per cubic yard	70 cents (wages).	
,, extracting, per cubic yard	$2.50	One day's labour.
Yield of one cubic yard of ground	$8.00	$5
Total yield of claim in one year	$18,500	$3000
Wages paid underground drifters	$2.50 per day.	
Number of men working	12	3
Composition of deposit from surface downward	Lava, or sedimentary formation; hard cement above ledge or bedrock; auriferous gravel-cement; hard quartz, black and dark coloured.	Lava, gravel—cement, fragments of slate, granite and gneiss, rounded, loose sand.
Direction of channel	N.W., S.E.	N.W., S.E.
Bedrock	Very hard, compact, gray quartz slate.	Slate, blue.

The following are the details of the Live Yankee claim.†

* Min. Com. Rept., 1872, p. 72.

† *Ibid.*, 1875, p. 153.

Length of tunnel	2600 ft.
Yield from 1855 to 1863	$698,534.00
Cost of opening and mining	$370,166.00
Total product of mine	$1,000,000.00
,, receipts to July, 1874...	$328,352.38
,, expenditures	$164,352.38
Number of carloads extracted	115,950.00
Paid per carload, one cubic yard	$2.76
,, square foot of ground	$1.09
Number of square feet washed	292,200.00
Superficial feet contained in the channel	7,232.00
Which, at nine cents per square foot, is equal to ...	$7,882,880.00

They used 4000 lb. to 6000 lb. of giant powder per year, 2000 lb. to 3000 lb. of steel, and 300 to 400 boxes of candles. There were eighty-five miners in the mine, at $3 per day; thirty cars, which are used for extracting the gravel which is dumped in the yards to be sluiced. The sluice is 6600 ft. long, 17 in. in width, and 18 in. high. It has a grade of 6 in. in 12 ft., and is paved with blocks 16 in. by 16 in. and 4 in. thick. The gold is so coarse, and settles so easily between the blocks, that no quicksilver is used. The head of water is 100 in. to 150 in.

At Gold Run, in the year 1872, a tunnel 1600 ft. long, with branches and gangways, was run for working cement. Twenty-five men at $3 per day were employed, who delivered between forty and fifty carloads in twenty-four hours, the quantity depending upon the nature of the ground. They worked a breast 110 ft. wide, taking 6 ft. to 7 ft. in height. The gravel was dumped directly into a stamp mill at the mouth of the tunnel. This mill had eight stamps of 500 lb. each, with a drop of 8 in. It was run by a hurdy-gurdy wheel 10 ft. in diameter, with a pressure of 80 ft. of water. The capacity of the mill was between thirty-five and ninety carloads, depending upon the rapidity with which the mine was worked.

The following Table gives the operations of the Indiana Hill, Blue Gravel Company from the date of incorporation, April, 1872, o the close of the mining season of 1874:*

* Min. Com. Rept., 1875, p. 100.

Mining Season.	Number of Carloads.	Yield per Load.	Total Yield.	Total Cost Mining and Milling.	Net Profit.
		$	$	$	$
1872	3,680	5.27	19,410.97	7,360.00	12,050.97
1872 to 1873	6,300	4.00	25,200.00	13,175.00	12,025.00
1873 to 1874	10,017	3.08	30,811.50	20.034.00	10,777.50
Total .	19,997	...	75,422.47	40,569.00	34,853.47

Recapitulation.

	$
Average yield per caload (19½ ft.)	3.77
,, ,, ton of 2000 lb.	4.71
,, ,, cubic yard	5.29
,, cost per carload (19½ cubic feet)	$2.03
,, ,, ton of 2000 lb.	2.54
,, ,, cubic yard...	2.90
Profit per carload	$1.74
,, cubic yard	2.39

It is estimated that the yield of the cement will be from $3 to $3.50 per cubic yard in cement mills. At French Coral the yield was $3.50, and the cost $1.10.

The following are the statistics of some of these mines in the Slate Creek basin channels at Holland Flat, in Potosi:*

——	Height of Ground.	Number of Square Feet.	Total Yield.	Yield per Square Yard.
			$	$
Down East Co. .	4 ft. 6 in.	287,000	346,000.00	10.80
Union Co . .	do.	1,300,000	1,187,284.74	8.10
Hawkeye . .	do.	348,000	365,819.40	9.45
Pittsburgh Co. .	do.	430,000	352,549.81	7.38
	Total .	2,365,000	2,251,653.95	

This gives an average of 95 cents per square foot of ground 4 ft. 6 in. high, and of $5.70 per cubic yard of gravel, which costs 47 cents per square foot, leaving a profit of 48 cents.

At Grass Flat, in Pioneer County, a branch tunnel, most of it 15 ft. above the bedrock, yielded:†

* Min. Com. Rept., 1875, p. 155.

† *Ibid.*, p. 156.

Square Foot of Tunnel.	Cubic Yards of Gravel.	Total Yield.	Per Linear Yard of Tunnel.	Per Square Yard of Tunnel.	Per Cubic Yard of Gravel.
		$	$	$	$
3000	475	384.52	2.3072	1.1536	0.8095

The main tunnel of the Pioneer Company's ground was run 1186 ft. in gravel from 10 ft. to 20 ft. above the bedrock:

Square Feet of Tunnel.	Cubic Yards of Gravel.	Total Yield.	Per Linear Yard of Tunnel.	Per Square Yard of Tunnel.	Per Cubic Yard of Gravel.
		$	$	$	$
6616	883.37	1,400.53	3.56	1.91	1.59

The following Table* gives the cost of drift mining in McAdam and Indian Creek Districts, Siskiyou County, California:

Cost per Cubic Yard Mined.	Cameron Mine.	Carrol Mine.	Hardscrabble Mine.	Lincoln, Hart and Henry, and Oak Grove Mines.	Siwash Mine.	Williams Mine.
	$	$	$	$	$	$
Labour . . .	0.952	1.148	1.148	1.148	0.66	0.552
Timber . . .	0.15	0.316	0.316	0.316	0.15	0.164
Powder . . .	...	...	...	0.005	...	...
Lights . . .	0.02	0.064	0.05	0.064	0.02	0.012
Falls . . .	...	...	...	...	...	0.036
Horse feed . . .	0.02	0.27	0.02	0.027	0.02	0.016
Incidentals and wear and tear . . .	0.065	0.074	0.12	0.074	0.065	0.11
Total cost of drift mining per cubic yard	1.207	1.629	1.50	1.634	0.915	0.89

These mines are shallow workings, from 28 ft. to 74 ft. in depth, the shafts penetrating a light wash and tailings. The "pay" is quite regular, lying on and in the bedrock, and is stoped out on the long-wall system, hoisted in buckets to the surface by horse whims, and washed in sluices near the shaft mouths.

* Report of the U.S. Census for 1880, vol. xiii., p. 204.

The Bald Mountain Mine of Sierra County, California, which has been worked in a systematic manner since 1872, may be considered as a representative mine of the drifting class. The company owns 7500 linear feet of the channel, of which about 4000 ft. had been worked up to the close of 1880. Up to August 1, 1880, there had been worked 3850 ft. of channel, yielding $1,788,000, or at the rate of $464 per linear foot, at an expense of $240 per foot. The yield was on an average of $2.17 per carload, or $1.75 per ton.

	$
The original outlay on shaft and tunnel was	20,000
Purchase of tailings, outlet and disputed title	31,000
Working expenses to August, 1880	923,000
Total outlay	$974,000

The work of this company is given in the Table below.*

Year Commences July 1.	Carloads.	Gross Yields.	Per Carload.	Dividends.
		$	$	$
1872 to 1875 . .	195,940	544,000.00	2.77	284,000.00
1875 ,, 1876 . .	100,080	296,341.76	2.96	150,000.00
1876 ,, 1877 . .	98,044	235,803.57	2.40	70,000.00
1877 ,, 1878 . .	106,160	269,755.00	2.54	120,000.00
1878 ,, 1879 . .	90,274	164,909.00	1.82	40,000.00
1879 ,, 1880 . .	86,378	188,892.40	2.18	60,000.00
1879 (Oregon Creek)	...	3,000.00		
	676,876	1,702,701.73	*a*2.17	724,000.00
July 1, 1880 . .	...	16,914.38	...	20,000.00
To August 1, 1880 .	...	1,728,616.11	...	744,000.00

a General average.

The following Tables give the product of some of the most important mines in California.†

Mine.	Locality.	Length of Claim.	Yield per Linear Foot.	Product.
		ft.	$	$
Hidden Treasure .	Placer County	10,560	294.00	*a*274,283.00
Bald Mountain .	Sierra ,,	7,500	464.00	*b*1,728,616.00
Hawkeye . .	,, ,,	800	437.50	*c*350,000.00
Empire . .	,, ,,	1,560	482.05	*c*752,000.00
Monumental .	,, ,,	1,040	312.00	*c*325,000.00
Pittsburg . .	,, ,,	860	506.00	*c*420,000.00
Union . . .	,, ,,	2,400	625.00	*c*1,500,000.00

a To May 1, 1880. *b* 1872 to August, 1880. *c* 1870 to 1880.

* Prod. Gold and Silver in the U.S., 1880, p. 324.

† Report of U.S. Census, 1880, vol. xiii., p. 303.

The following Table gives the product and rate per foot of channal for ten years of some of the leading mines of Sierra County, California:*

Mine.	Length of Location.	Yield per Foot.	Total Yield.
		$	$
Union	2400	625.00	1,500,000
Hawkeye	800	437.50	350,000
Pittsburgh	860	506.00	420,000
Monumental	1040	312.00	325,000
Empire	1560	482.05	752,000

An analysis of the cost of mining in a series of mines in McAdams and Indian Creek districts is appended. These mines are shallow workings, from 28 ft. to 74 ft. in depth, the shafts penetrating a light wash and tailings. The "pay" is quite regular, lying on and in the bedrock, and is stoped out on the long-wall system, hoisted in buckets to the surface by horse whims, and washed in sluices near the shaft mouths.†

Cost per Cubic Yard Mined.	Cameron Mine.	Carrol Mine.	Hardscrabble Mine.	Lincoln, Hart and Henry and Oak Grove Mines.	Siwash Mine.	Williams Mine.
	$	$	$	$	$	$
Labour	0.952	1.148	1.00	1.148	0.66	0.552
Timber	0.15	0.316	0.31	0.316	0.15	0.164
Powder	...	...	...	0.005		
Lights	0.02	0.064	0.05	0.064	0.02	0.012
Falls	...	...	...	...	...	0.036
Horse feed	0.02	0.027	0.02	0.027	0.02	0.016
Incidentals and wear and tear	0.065	0.074	0.12	0.074	0.065	0.11
Total cost of drift mining per cubic yard	1.207	1.629	1.50	1.634	0.915	0.89

The total outlay of the Ruby Mining Company from January, 1880, to April, 1883, was:‡

* Prod. Gold and Silver in the U.S., 1880, p. 325.

† Report of the U.S. Census for 1880, vol. xiii., p. 204.

‡ Prod. Gold and Silver in the U.S., 1883, p. 718.

	$
Main tunnel, construction	62,874.36
Building the roads, flumes, &c.	11,751.23
General expenses	9,404.57
Prospecting work	8,203.14

Up to April 22, 1883, there were 2400 ft. of main tunnel driven through solid rock, 11,217 ft. of drifts in the channel, and 150 ft. of raises and winzes. Nearly the whole of this work had to be done before the mine commenced to yield a profit.

The expenses of this mine from April 16 to April 22, 1883,* were :

Labour, fifty men at $3		$150
Outside men and management		25
Powder, fuze, caps, steel, iron, light, timber and incidentals		29
Or, per day	$204	
A week of six days	$1224	
Running expenses on Sunday	30	
Running expenses for seven days	$1254	

The result was :

Carloads of pay dirt	709	
,, ,, waste dirt	200	
Total carloads	909	
The cost per carload was	$1.34	
The yield was 161 oz. in the head boxes cleaned-up, at $17.50 an ounce, equal to		$2817.50
Lower boxes not cleaned up, estimated at one-seventh of the head box		402.50
Total		$3220.00
The pay per carload was		3.64
Net profit for week		19.66

This company adjoins the Bald Mountain Company.

The Hidden Treasure Company, on the Forest Hill Divide, in Placer County, owns 10,560 ft. of channel. The total expenses from 1878 to 1880 varied from 43 to $67\frac{3}{10}$ per cent. of the product. The contract price for running the tunnel in soft slate, requiring but little blasting, the timbers and track being furnished, is $3.50 per foot. The total expense, including the timbers and iron rails, is about $4 per foot. From this tunnel upraises

* Prod. Gold and Silver in the U.S., 1883, p. 718.

are made every 125 ft. to the gravel, which is blocked off in squares of that length of face. The channel is worked for a width of 500 ft. The pay gravel is 5 ft. to 6 ft. deep, and consists of quartz, pebbles, and boulders, the latter sometimes being very large. The gold is 922 and 928 fine, and ranges in size from pieces of five to six ounces down. About sixty men are employed as miners, who are paid $2.50 per day. All the wood used is cut by contract during the summer, eight or nine men being employed for the purpose. It is shot down from the top of the ravine 500 ft. above the mine in a chute 1300 ft. long. Twelve thousand timbers of all kinds and 80,000 ft. of lagging are annually used. From 150 to 200 cars of gravel, or about 3500 lb., are taken out of the mine daily. Ten men were formerly necessary to run the cars and do the same work which is now done by two horses and two men. One man is occupied washing the gravel, and several blacksmiths and carpenters are employed. The value of the gold taken out in 1880 was $115,000. The cost of doing it, for labour and superintendence, was $50,000; all other expenses $8000; leaving a net profit of $57,000. The output in 1887 was $91,000.

At the Mountain Gate Mine the number of men employed is forty-five; thirty-six miners, two men at the water-wheel, two carmen, two blacksmiths, one carpenter, one gravel washer, one superintendent.

The details of the cost of working at the Mabel Mine is given in the Table on next page.

Owing to its peculiar nature very little improvement has been made in this method of mining since it was first invented. Drift mining may be said to be confined almost exclusively to California, and to have been developed there very slowly. It is on this account that so much less definite information has been published with regard to it than with regard to any other method of extracting gold in that country. Of all the methods of mining gold it has probably been less used and less studied. It is, however, more or less used in all the gold-mining regions of the State, and is likely in the future to attract more attention than ever. Such mines will always be numerous wherever the lava-capped channels are found. The great difficulty about its

COST OF DRIFT MINING AT THE MABEL MINE.*

Month.	Gravel.	Rock.	Miners.	Carmen, &c.	Total Men.	Powder.	Fuze.	Caps.	Candles.	Timbers.	Lagging.	B. Caps.	Tons Gravel per Man.	Labour per Ton.	Powder per Ton.	Fuze per Ton.	Caps per Ton.	Candles per Ton.	Timbers per Ton.	Supts. per Ton.	Cost of Extracting per Ton.	Washing per Ton.	Value per Ton.
1885						dols.		dols.	dols.	dols.	dols.	dols.											
August ..	168	18	43	16	59	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	.56
September	1386	153½	256	145	431	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	.56
October ..	2379½	364½	486	166	652	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	2.15
November..	3348½	185½	766	292	1058	22	..	9.75	..	105.85	96.15	28.64	..	..	.065	..	.0029	..	..	..	..	..	2.78
December ..	2830½	½	777	270	1047	221	..	12.75	72.00	..	..	..	..	..	.078	..	.0045	.025	..	..	..	..	2.05
1886																							
January ..	2913	147½	1016	306	1322	286	..	19.50	76.80	139.02	106.15	34.24	..	..	.098	..	.0066	.026	..	..	..	..	2.13
February ..	1478½	21½	388	147	535	..	..	..	..	101.70	73.20	14.64	..	..	..	..	..	..	..	..	..	..	2.30
March ..	134½	..	24	10	34	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	2.30
April ..																							
May.. ..	894½	295½	187	110	297	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	1.74
June ..	1234½	420½	158	105	263	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	2.42
July ..	2322	819½	776	346	1122	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..	3.99
August ..	1746	387½	405	183	589	..	..	..	..	..	..	..	..	1.39	..	..	..	..	..	.132	..	.074	2.51
September	3320½	461½	1155	263	1418	364	20.80	22.50	62.40	78.87	74.80	28.80	2.34	1.04	.109	.0062	.0067	.018	.054	.099	1.333	.041	2.07
October ..	3710¼	437½	1238	301	1539	468	30.00	27.75	76.80	68.92	60.95	11.52	2.41	0.96	.126	.0060	.0074	.020	.038	.083	1.242	.037	1.76
November..	3216	375	1090	251	1341	390	24.40	23.25	57.60	75.27	51.97	4.80	2.39	1.27	.121	.0075	.0072	.017	.041	.103	1.567	.040	1.63
December ..	3286½	406½	1128	248	1376	507	29.00	23.75	72 00	79.90	52.65	7.36	2.38	1.21	.154	.0083	.0072	.021	.042	.099	1.542	.039	2.05
1887																							
January ..	3943½	763½	1105½	378	1487½	533	33.60	25.50	76 80	75.42	43.87	8.00	2.6	1.00	.135	.0085	.0064	.019	.032	.087	1.288	.037	1.56
February†..	1299	319½	399	209½	609½	156	9.60	6.75	33.60	26.65	16.12	3 44	2.13	1.78	.12	.0073	.0057	.025	.035	.235	2.207	.081	2.05
March ..	3349½	174	949	423½	1373	312	29.20	21.00	57.60	100.30	56.50	30.80	2.80	.95	.081	.0075	.0054	.014	.048	.089	1.195	.038	1.64
April ..	4308	84½	969½	350	1309½	338	28.00	21.00	62.40	106.12	96.10	40.16	3.29	.85	.078	.0064	.0048	.014	056	.077	1.086	.031	1.59
May.. ..	4017	6½	983½	360	1242½	364	28.00	23.25	81.60	104.07	93.02	38.96	3.23	.91	.09	.0069	.0057	.023	.058	.085	1.179	.035	1.68
June ..	3832½	..	893	397	1290	312	24.00	24.75	57.60	98.95	90.72	35.44	2 97	.90	.08	.0062	.0064	.015	.050	.083	1.147	.036	1.24
July ..	3417½	10½	807½	326	1133½	312	28.80	25.50	57.60	85.05	75.60	26.32	3.01	.98	.093	.0084	.0074	.016	.054	.099	1.258	.037	1.47
August ..	3063	141	..	..	1005	364	23.20	23.25	48.00	88.10	74.10	28.08	3.01	.89	.118	.0076	.0075	.015	.062	.112	1.21	.041	2.13

* This Table was prepared by my former pupil, W. H. Radford, of N. Bloomfield, California.
† Heavy storm.

development is the difficulty of finding the channel. Hundreds of tunnels, from a few hundred feet to several thousand feet in length, have been run too high to get at the rich gravel, or have missed finding it at all. On this account it is too uncertain to attract any very large amount of capital. It is not now carried on as extensively as during the period from 1855 to 1870, a large part of the easily accessible and best-explored ground having been worked out. A great deal of the ground explored has been found too poor to work. At the present time Sierra, Nevada, Placer and Plumas Counties contain the most of the mines. The decision of the Courts on the débris question has re-directed attention to this method, and its importance is daily increasing. About one-third of the gold credited to deep placer mining is now produced by this method, which is in use to some extent throughout the gold-bearing region of Central California, but only on a large scale in Placer and Sierra Counties, where the channels are covered to great depths. It is practised to a very limited extent in some of the other Territories, but has not the same attractions there, or the same necessity for existence that it has in California. As we have seen, some of the California mines have yielded very large quantities of gold, but, as in every other case, the successes have been heralded everywhere, while the failures are known only within a very contracted area. It is, however, true that this kind of mining will yield, with the same amount of intelligence expended upon it, as fair returns for the capital invested as any of the other methods of gold mining.

The construction of long bedrock tunnels at points where the channel has been demonstrated can always be done without risk. The width of the Blue Lead channel varies from 300 ft. to 600 ft. The portion which will pay by the drifting system is rarely narrower than 180 ft., and often wider than 300 ft. The cost of mining and washing is about $1.33 per carload, and the yield varies from $2 to $5 per carload. At Bald Mountain it is 500 ft. wide. If 200 ft. is a fair average width of the pay gravel, and 4½ ft. an average of height of the pay, there would be 165,353 cubic yards of ground for 5000 ft. of location, equal to 230,706 carloads of loose dirt. At $2.10 per carload, which is somewhat under the value of gravel in this district, it would give a gross

yield of something over half a million dollars for the 5000 linear feet of channel. The usual ratio of profit is about 50 per cent. The working life of mines of this kind is from six to twenty years, or rarely over thirty, depending upon the length of the channel, the facilities for taking out the gravel, and the number of men employed. When a number of claims have been treated on a well-defined channel which has been thoroughly explored, the claims are sometimes worked in rotation, the lowest one working first. When their ground is exhausted the next company works its ground, either by opening through a new tunnel, or by acquiring the right of way from the company who worked out the ground below. Occasionally there is no other way of working a claim except through the tunnel of the claim below. The number of men employed varies with the difficulty of breaking the ground, the quantity of water encountered in the working, the season of the year and the size of the claim. It varies from 10 to 15 to 150 and occasionally as high as 250 miners in each mine. The immense amount of country underlying the volcanic formations will furnish large quantities of gold for many years, and it will undoubtedly be one of the chief sources of gold supply in the future history of California.

When a drift mine not capped with rock is worked out and falls in, the ground above is broken up and is better fitted for hydraulicking should this process ever come into use again. The extraction of the very rich deposit on the bedrock has served to break up the gravel above and render it fit for piping without the necessity of blasting.

This method resulted from necessity and not from choice. All these claims are situated in the higher ranges of the Sierra Nevadas, and are sometimes covered with hundreds of feet of basalt volcanic mud and scoria. At Bald Mountain it is capped with a table of basalt 100 ft. or more in thickness. The process of drifting is the only one available for the extraction of the gold, and, the cost being from $1 to $2 per cubic yard, only the lower stratum can be profitably worked. The ordinary section of a drift claim would differ in no respect from that of an hydraulic bank, except when volcanic matter covers the auriferous gravel. As

in the hydraulic mining, the principal factor in the successful operation of these claims is either the ownership or the control of an abundant water supply at all seasons from a source sufficiently elevated to command large areas of ground, and a market for the sale of the water which is not required for their own use. Ordinarily, however, the drift companies do not own their own water; they purchase it.

Treatment of Tails.

The tails from the hydraulic mines in Indian Cañon, at Iowa Hill, have accumulated until they are now in places 75 ft. deep, and are so high that occasionally they have run out the grade of some of the hydraulic mines. The Indian river runs in the summer a small stream not over 2 ft. wide and a few inches deep, but in the winter and spring it is a large and swollen stream. A patent, three miles long, is owned on this river, and they are now working over the débris of the last thirty years of mining. This is done in a flume, which is working at the lower end and advances up stream as the lower part is washed out. It is intended to work out the tails of the whole cañon. No piping is done, but the water of the river is depended on to do the work. To throw the whole of the river into the flume, wing dams are constructed. Generally the water of the river is sufficient, but when it gets too low men are obliged to shovel in the gravel. The cañon worked is on an average 300 ft. wide. The flume is 12 ft. wide, and divided into two longitudinal compartments, both of which are paved with stone. One is 8 ft. wide and the other 4 ft. Both are constantly open. The larger one is destined for very heavy water and for stray logs, but both remain constantly open. The head boxes of both compartments are fitted with railroad iron, so as to prevent excessive wear. These protected compartments are moved to the next head as the flume is extended up stream. Three-quarters of a mile of this flume has already been constructed, and they are now going up stream with it. When the river is no longer sufficient to carry the gravel into the sluices, they are closed up, and when this is done the bedrock is cleaned, as is usual in cleaning bedrock, with small sluices. There are no derricks used. When the

boulders are larger than two men can handle they are blasted. The stones are usually small in size. At the end of the flume there is a drop of 60 ft. to 70 ft., which gives plenty of dump for the tails. The river empties into Shirt-Tail Cañon, which empties into the north fork of the American river. The whole work of the flume requires only five or six men. In the summer, when the river is dry, only one man is kept as a watchman. It takes five to six weeks for the men to clean the flume and the bedrock. There is plenty of water for the work of the flume during four to five months of the year. When it commences to get scarce, the men help the movement of the gravel with shovels.

The work of the flume is carried on during eight months of the year, and the enterprise so far has paid well. Other claims are owned above this, but they cannot work until this one has been worked out, as there is no dump room for them.

At Hunt's Placer claim, Snake River, near Bliss's Station and Salmon Falls, in Idaho,* there is a placer claim where the deposit is on lava and the gold very fine, supposed to have come from the decomposition of the volcanic material. To work this a ground sluice is prepared, into which the pay dirt is both thrown and hydraulicked. This sluice ends in a "machine." This consists of a rectangular box of wood, 15 ft. long and 5 ft. wide. The bottom inclines at both sides to the centre at a low angle, so that it is 3 ft. deep in the centre. This box is covered with an iron plate, $\frac{3}{16}$ in. thick, which is filled with $\frac{1}{8}$ in. to $\frac{3}{16}$ in. holes, which are conical with the larger diameter down. The bottom of this sluice ends in a head box at right angles to the machine. The head box has gates for five low-angled sluices, 22 in. wide and 30 ft. long, covered with burlap held in by slats. The gravel runs on to the iron plate, which allows the fine to pass, and acts as a grizzly or sieve. The fine ore and sand goes through to the burlap. The fine gold collects on the burlap, which is rolled up one way on the machine and unrolled the other in the water tanks every hour to take the gold out. When the burlaps get slimy they must be dried in the sun, in order to kill the vegetable growth which would otherwise destroy them.

* For this information I am indebted to my former pupil, W. P. Jenny.

They are washed for the gold every other day. The quantity collected is not large, but sufficient to pay well. The sands are taken up in boxes lined with tin, and the burlaps washed in a special tank. The sands from the boxes are treated on the rocker without chemicals. Those from the tank are first thoroughly wet with a solution of 4 oz. of potassium cyanide, in water enough to wet down the whole of them. This is allowed to remain over night. The next morning it is treated on a rocker 5 ft. long, with an amalgamated plate the whole length of the bottom of the rocker, which catches the gold, so that the tails do not run over $8 to the ton. The gravel yields about 8 cents in gold to the cubic yard. It is concentrated in passing the grizzly. It is this material, which is composed of the black iron sands, fine stuff, and gold, which goes to the burlaps. The gold colours, which ordinarily in California are counted as 10 cents, are here worth only a small fraction of a cent. Many of them are so light that they float on the water, and are consequently very difficult to collect. There are three of these machines working. The labour required is very small. There is one man only in the pit, who regulates the flow of the water and directs the current by boards 12 ft. long, 1 ft. wide, and 1½ in. thick. These boards have a round stick in the middle projecting on each side, so as to move them easily. At the clean up the amalgamator and the pitman work together. There are only five men for the three machines, one of whom does other work. When the burlap has been left on the sluices for some time it is found yellow with gold, but it is so very light that what appears to be a large quantity is of little value. The process is very effective and cheap. The returns are about $50 a day, of which $25 to $30 are clear profit.

This process with burlaps resembles one in Japan, where matting serves to collect the gold, but no chemical is used. The same process of folding one way and unfolding the other is also found to be necessary to separate the fine gold caught in the matting.

Note.—A complete description of this process, with drawings of the machine, will be found in the Trans. Am. Inst. Mining Engineers for the Ottawa meeting, October, 1889.

CHAPTER IV.

TREATMENT OF GOLD IN THE ARRASTRA.

THE treatment of silver in the Patio process, and the treatment of gold in the Arrastra in so far as native gold was found with the silver ores, and was, therefore, collected in the same process, but in a preliminary operation, has been previously discussed.* The treatment of certain kinds of gold ores by this apparatus is of great interest at the present time, when this very effective, but much misunderstood machine, is being extensively copied in principle in the new methods which are being devised for the treatment of gold ores. The machine is certainly a very primitive one, but its use in the early days of mining in California was very remunerative, as all the ores were free milling. Even after the sulphurets began to appear it held its own for high-grade ores, and it was only when it was ascertained that low-grade ores could be treated that it fell into disuse. It does not appear that the South American mills† were used on this continent.

Notwithstanding the extreme slowness with which it works, and the rapidity of the stamp-mill and the pan, it has not been entirely replaced by them, and can often be used where, from local circumstances, the nature of the ore, or the limited amount of capital at the disposal of the miner, neither the stamp nor the pan and settler could be used. The idea of modern amalgamation with the pan was undoubtedly taken from the arrastra, which the early settlers of California found in use among the Mexicans when they went to that State. It combines, although a very simple and often a very rude appliance, most of the best points of the stamp, the pan and the settler. But to the early miners the advantages of its simplicity of construction, its cheapness and its high yield, were as nothing compared with the necessity of

* Chap. vi., vol. i.

† School of mines Quarterly, vol. viii., p. 61.

making a large output in a short time, whatever the loss, and they thus modified it little by little, trying to keep its essential features and principles, until it disappeared so entirely that they lost sight even of the fact that it had ever been used. It has real advantages, however, which prevented its being absolutely relegated to the catalogue of historical metallurgical appliances. Its cheapness and simplicity of construction place it easily within the means of those who have but little capital to invest, and, therefore, cannot erect a stamp-mill while prospecting, or even developing their claims. The economy with which it can be erected and worked, and the high yield which it gives of the precious metals when it is properly managed, more than counterbalance the deficiency of output, especially when the claims are very small, not rich, or where they are not certain to last for any great length of time. It is especially adapted to the class of free-milling gold ores which are generally found on the outcrops of veins or near the surface. Other classes of ores which are not free-milling are not suited to it, or to any other process which does not require a much more expensive plant. Thirty arrastras run by water-power were, up to 1884, used at the Sierra Buttes Mine for working the concentrates rich in gold. Their place since that time has been taken by Frue vanners and a chlorination works, but tails are still worked in them there. Possibly the future may develop a leaching process for gold, as it has done for silver, which will be its equal, or perhaps its superior; but this is in the future, and for gold ores has not yet been realised, except where small quantities of gold ore are found in large quantities of silver. These facts have been so well recognised within the last few years that there are now many works, situated principally in the Western States and Territories, that are successfully working arrastras, which are giving a high yield. A very large number of such works have been erected since 1876. Some few antedate that time. The Accidental Mine,* for example, in 1863 and 1864, treated in the arrastra 1000 tons of ore yielding $25 to $75 to the ton. The Sexton Mine in the same year treated ore which averaged $37, and three tons of it gave $191 to the ton. Many of these works

* Lock on Gold, London, 1882, p. 178.

have been built as a makeshift to take advantage of some peculiar conditions which were known at the outset to be only of a temporary nature. Some few of them have been altered from abandoned pan-mills, but most of them have been constructed especially to treat the ores of small but productive mines, which are too small to be worked by companies. Occasionally they are built in districts where only small mines are found, as custom-mills, and without any mining property attached to them. Others, as at Sierra Buttes, have been built to treat the tails of old mills, which can often be done with large profit, or even to work up poor tails, when free water power can be had. A number of such mills are at present working on Comstock tails. When steam is to be used the tails must be correspondingly rich, enough to pay for the expense of the fuel used to get the power. As the life of such works is dependent on circumstances which are almost accidental, it is usually very short, for which reason they are rarely built by companies with much capital, but they serve very useful purposes at small mines worked by the farmers in the intervals of their business, and for working tails where the means at the disposal of the miner are very limited. They are employed to a large extent as clean-up pans in the hydraulic and drift mines, and also in working free-milling gold ores. They are also used for treating silver ores containing gold which produces a doré bullion. They are not generally used on high-grade ores, as the pan is much quicker for such ores as these. Very little ore, averaging as low as $10 a ton, is treated in them. Generally the yield of the ore must be much higher. They do not differ essentially in construction from those which have already been described,* but they have been slightly modified from the Mexican original.

When constructed for permanent use they are built with a circular pavement of rock 6 ft. to 10 ft. in diameter. This pavement is made of the hardest and toughest rock that can be found in the country laid on a bed of well-puddled clay from 3 in. to 6 in. thick. The rock should in preference be coarse rather than fine grained, and should remain rough after long use, not

* Vol. i., chap. vi.

becoming polished by the grinding action, as this would tend to allow the ore to slip around rather than to be acted upon. It should also have the power of rubbing the fine particles of metal bright, so that they will be quickly attacked by the mercury. The rock which is used for this purpose is generally granite or basalt, or some of the various kinds of compact quartz which are neither agates nor chalcedony, but intermediate between the two, which are so common in the west. When new this pavement is made at least 12 in. thick. The stones used must be as large as possible; they are generally jointed with great care and set in hydraulic cement, if it can be had, in order to prevent any leakage of mercury or amalgam, which will settle in the crevices of the stones, from which it may be picked, but it should never go into the ground beneath. When they are small, and even at times in large ones, the pavement is not so carefully made, and the stones are set with the joints only apparently closed. In the centre of this pavement a pivot is fixed for an upright post, which is supported overhead, and which carries a little above the level of the pavement from two to four arms. To these arms, stone drags, weighing generally from 200 lb. to 1000 lb. or even 2000 lb. made of the same material as the pavement, are attached by ropes or chains. They must be cut so that the centre of gravity falls below the centre of the stone; they must also have a broad flat surface on the bottom to keep them from turning over. They are hung so as to be raised a little above the pavement in front, but they drag behind. One of them is hung so that the outside edge works in advance of the inside, so as to throw the charge to the centre. The other is set exactly the reverse. This disposition keeps the charge thoroughly mixed. The arms carry sometimes a single drag only at their ends; they occasionally have two, but rarely more. The number of drags in each arrastra varies; when it is difficult to get stones of sufficient size, from two to as many as twelve are used; four is, however, the usual number when the stones are large. They are sometimes as light as 80 lb.; they will average, however, at least 300 lb. when new; occasionally they are made as heavy as a ton. The arms to which they are attached are moved by horses or mules, or by water power applied in the

ordinary way, or by horizontal wheels around the outside of the arrastra, or by hurdy-gurdy wheels, or by steam. Their speed varies four to eighteen turns per minute. When power is used it will average from ten to fourteen turns per minute. When the arms of a 10-ft. arrastra make fourteen turns the outer drag has a velocity of about 400 ft. per minute. The average, however, will not be higher than from 200 ft. to 300 ft. per minute. The outside of the pavement is protected by a vertical wall made of flat stone or of plank driven into the ground, but made tight both on the inside and outside. It is from 18 in. to 30 in. above the pavement, according to the size of the arrastra. These planks or stones keep the pulp from spreading and fix the depth of the arrastra. In these sides there are either a series of round holes closed by wooden plugs like those in the common settler, or a gate on the side which is used to empty it.

The old tails of the Sierra Buttes mines have been leased and are now being worked in arrastras, of which there are five. Four of these tail into the fifth. These are 12 ft. to 16 ft. in diameter and 4 ft. 6 in. in depth. They use from four to eight drags, which usually weigh about 1000 lb., but some of the larger ones use those weighing as high as a ton. When such large drags as these are used, smaller ones are always put in between them on the arms supporting the water-wheel which could not carry such a weight. All these arrastras are run by a wheel around the outside of the arrastra, supported on four beams at right angles to the drag beams. The arrastra is built in two ways. Staves 2 ft. thick are driven into the ground close together. Around these a wall of stone is built up 2 ft. in thickness, bound in wood and hooped. Or the arrastra is built entirely of stone and lined with it and held with wood and iron bands on the outside. They make twelve revolutions a minute. The drags are fastened to the arms by ropes tied to wooden pegs driven into holes drilled in the stone. They usually have only two supports. The drags are always made just as large as they can put them in. They are a hard-grained quartzose conglomerate. When the drags are smaller they put in just as many as can be hung to all the arms, both the drag arms and

those which support the water-wheels. The tails are fed into the arrastra, which are constantly turning, and when they have been worked a sufficient length of time water is turned in and the excess let out at the top and generally made to tail through to one below. They are never cleaned up until the stone bottom makes it necessary, so that they may run five months without stopping at all. The bottoms are made, as in the small arrastra described, below the pavement stones and are not put in very close nor with cement. The stones are simply laid in, and when a clean-up is to be made they are all taken up and the contents washed down to the black sand. It has been found that wooden linings for the sides work just as well and last just as long as stone and are much cheaper. The water power is taken in flumes from the river. The space that these arrastras occupy is quite confined, so that there is not all the grade there should be to work them to the best advantage. The cone to support the upright in the centre rises 2 ft. above the bottom, and the pin which serves for the rotation of the axis of the upright is fastened on to the outside of the transverse beam, so that the axis can be moved easily. The arrastras are all placed together without systematic arrangement but wherever sufficient room could be found to put them in.

The arrastra is used by the small farmers near San Andreas, in California, on what are called poor mines; that is, mines which are small, which pay to work but which are not worth the attention of a company. They are worked by the farmers, who find the mines and who use their stock and labour at odd times. They are built in a circle made 7 ft. in diameter and are constructed with staves 2 ft. long, 2 in. thick, and 3 in. wide, driven into the ground as close as possible and secured by two iron hoops. The outside ground is about 1 ft. from the top. The bottom of the arrastra is thus below the level of the ground. In the centre is a step made of wood and bound together by iron hoops, in the same way as the outside of the arrastra; on this the centre pivot is placed. This is fixed in a wooden post 9 in. square, whose top pivot is placed upon the outside of the overhead beam, so as to be easily reached. On this upright an arm is placed for the drags, of which in such arrastras there is only one at the end of

each arm. They are rough quartz and are hung with chains in such a way that the hind end drags. This drag arm projects 1 ft. The mule arm on the mule side projects 4 ft. 6 in. and is made of a beam 4 in. by 6 in. with the 6 in. side up. On this side the whiffletree is placed. The other end projects only 1 ft. To make the horse go around, a piece 2 ft. above the mule arm is nailed on to the upright, and to the end of this a pole about 4 ft. long is attached by a swivel joint made by a string or thong. To the other end is a snaffle to attach to the bit of the horse. This always forces him to move in the same direction and around the outside of the arrastra. The bottom of the arrastra is made by putting in different coloured sands with generally black on the bottom. This is run upon until it is packed. Another coloured sand is then put in, and then the stones, which are ground flat. This leaves a considerable interval between the stones. About 1 lb. of mercury is used in the arrastra. The clean-up is only made when the pavement is worn out. They then dig down to the black sand, washing all the bottom, and put in another bottom in the same way. The water is brought to them by a 1½ in. pipe on one side through a small opening, and the tails sluiced out through a gate which is 6 in. by 4 in. nearly opposite. The tails run into a sluice 1 ft. square and 12 ft. long, with circular riffles, and then over blanket sluices 12 ft. long and 1 ft. wide. From here they run to waste. Very little is caught in these sluices. Most of the gold is collected in the arrastra.

The difficulty of getting the stone for the pavement, and of properly constructing it, has caused the invention of an iron arrastra,* which is really a double pan. It varies from 6 ft. to 8 ft. 6 in. in diameter. The central portion of this machine is a pan, the muller of which is used for grinding. On the outside of this there is a circular space 14 in. wide and 5 in. below the muller of the pan, in which a set of drags travel. It is fitted with a revolving amalgamated copper circle, which acts like the apron of gold mills. It is run by power from below like the pan. Such an arrastra, 8 ft. 6 in. in diameter, requires five horse-power to run it, and can treat from 7 to 10 tons of ore in

* Invented by A. B. Paul, of San Francisco.

twenty-four hours, the amount depending upon the fineness of the ore treated and its hardness. The whole machine weighs about 4½ tons, and requires five horse-power to move it. The ore to be treated is first crushed through a 16 to 20 screen. It is either charged by hand or is sluiced into the machine. As fast as the ore is reduced fine enough it is carried by the water into the arrastra. To insure an even distribution, it is usually charged in two or three places at a time. The supply is thus made perfectly constant. The drags of the arrastra weigh 220 lb. and are so hung that they ride the ore instead of pushing it forward only. The quicksilver is introduced just as it is in the Mexican arrastra, the quantity depending on the richness of the ore. In the bottom of the arrastra circle there is a groove to which an amalgam safe is attached. When the amalgam is caught, the free mercury travels on. When the sulphurets accumulate in such quantities as to hinder the work, they are washed out by drawing a plug, and then put to one side to accumulate until there is sufficient for further treatment. This arrastra is generally used below the stamps.

Another and much simpler machine is shown in Fig. 40. It

Fig. 40. Prospect arrastra.

consists of a wrought-iron pan, 3 ft. in diameter and 12 in. deep. The top edge is reinforced with a hoop rivetted on to it. The lower edge of the rim is rivetted to the bottom. An axis passing through a tube rivetted to the bottom by a flange carries a pulley below, on which a belt to communicate the power used is run. The upper part of this axis carries a yoke, to which drags are attached by chains. When it is to be used as an arrastra, as shown in Fig. 40, it is provided with a steel bottom, on which the drags run.

When it is to be used for amalgamation, the drags are removed and another yoke is substituted, shown in Fig. 41, and to this copper fingers are attached. This, when the steel bottom is removed, makes a very good clean-up pan. Such arrastras as this are very much used in prospecting or even in the regular working of small claims. They are very light, the heaviest single piece weighing not over 150 lb., so that they admit of easy transportation. In many of the hydraulic and drift mines the clean-up pans are arrastras made of iron. They vary from 4 ft. to 7 ft. in diameter, and usually have cast-iron drags, as the use of hot water in winter is apt to crack those made of stone. The arrastra used at the Mabel Mine is 4 ft. in diameter, 22 in. deep, and has two cast-iron drags which weigh 200 lb. each. At the Derbec Mine the diameter is 6 ft. 6 in., the depth 30 in., and

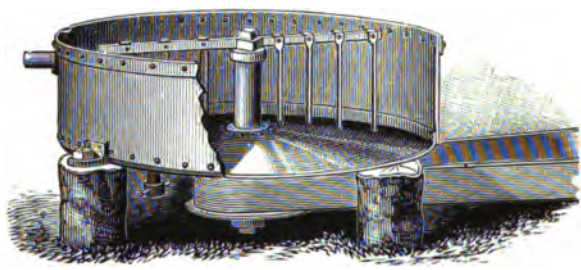


FIG. 41. PROSPECT CLEAN-UP PAN.

the weight of the two drags 200 lb. each. The greatest care must be used to keep oil and grease of every kind out of these machines.

The usual arrangement of the works is to have the arrastra a little below the level of, but directly in front of, the Chilian mill, rock breaker, or stamps, and so near to it that there will have to be very little handling of the broken ore to reach the arrastra. The power is generally transmitted to the drags by a gear-wheel above and by a belt from the source of power. Where horses or mules are used they are generally hitched to the extension of the arms which carry the drags. A 10-ft. arrastra will require two mules.

To treat the ore in the arrastra, it is first broken to the proper size. This is sometimes done by hand, but oftener by a Chilian mill, a crusher, or by stamps. When stamps are used in connec-

tion with the arrastra, as they frequently are, either when a pan mill has been abandoned or its operations changed, so that certain classes of ore are treated in the pans and others in the arrastra, the motars are set without screens and the ore crushed dry. The ore is fed as usual, but it is delivered by the blow of the stamp, just as it strikes it, so that it varies in size from dust to the size of a pigeon's egg. It is charged into the arrastra just as it comes from the stamp. A battery of five stamps will crush, in two hours, all the ore that a single arrastra can treat in twenty-four hours. Such a battery, therefore, will be capable of doing the work for twelve arrastras. It is always desirable to use either a crusher or a stamp, as it is more economical to break the ore in this way than to wear out the pavement of the arrastra in doing the same work. The crushed ore is shovelled or fed into the arrastra, and sufficient water added to make a thick pulp. A charge is generally from 600 lb. to 1000 lb. of dry ore. The arms are turned so that the drags act from two to four hours. The limit of time is fixed by testing the fineness of the pulp between the thumb and fore finger. The practice varies with regard to the addition of the chemicals, and also as to the chemicals used; in some mills they are added at the commencement, as soon as the drags are in movement; in others they are added at the same time with the mercury. The chemicals used with gold ores are a little potassium cyanide to "liven" the "quick," or some wood ashes or lye to neutralise the effect of any grease which may have got into the ore. Generally, no other chemicals are used. When the ores carry large amounts of silver, copper sulphate (bluestone) and salt, which are put in solid, are used in such quantities as the amalgamator thinks best, or the assay made by panning shows to be necessary. When the pulp has been ground long enough the mercury is added, the quantity depending upon the precious metal contents of the ore. It varies in amount, generally from 100 lb. to 150 lb. With high grade ores it is often from 200 lb. to 250 lb. Treating silver ores, or ores in which the silver is in excess, the amount of mercury added is large, often amounting to a pound of mercury or more for every ounce of silver. This makes a very liquid amalgam. But in treating gold ores, where the object is to

produce rather a pasty amalgam, the amount is much less. The condition of the pulp is judged of by sampling and panning it. Fire assays are not often made. When the assay shows that most of the precious metal has been caught by the mercury, the pulp is thinned with water and the speed of the drags reduced, in order to allow the amalgam to settle. After a time the arrastra is discharged, either by the holes in the side or by the gate. The time required for the operation is from six to ten hours. The capacity of the simplest arrastra varies with the kind of treatment and the completeness of the plant, from one to two tons a day. When the mill has rock breakers and stamps, so that it has but little grinding to do, the capacity depends upon the time necessary to complete the amalgamation. A 12-ft. arrastra driven by power, with heavy drags, making fifteen revolutions per minute, may treat two charges of two tons of ordinary ore in twenty-four hours, if very close work is not necessary, but this is the extreme limit of its capacity. When silver ores are treated the arrastra is often used only for grinding; the pulp is discharged into the pans and treated as in pan amalgamation, except that no grinding is done. When gold ores alone are treated the pulp is either washed into a settler, and the amalgam collected there, or the pulp is washed out of the arrastra and the amalgam collected in it. The Table below* shows the amount of ore treated in 1880 by twenty-two arrastra works, with fifty-three arrastras, which were not all running continuously. The average per mill was 413.75, and the average for each arrastra 171.70 tons.

State or Territory.	Number of Works.	Number of Arrastras.	Tons Treated.
Arizona	4	9	195
California	4	13	3205
Colorado	2	6	284
Idaho	2	4	2022¼
Montana	3	5	2250
New Mexico	1	4	500
Oregon	6	12	645
Total	22	53	9101¼

* Report of the U.S. Census, 1880, vol. xiii., p. 283.

When custom ore is treated, a clean-up is made for each run. It frequently happens that the amount of ore treated is very small, sometimes as low as half a ton when the ore contains from $500 to $600 to the ton or richer, so that the clean-ups have to be very frequently made. When the work is done on the ore owned by the works, it is generally made only once in two weeks. The amalgam is strained by hand in a canvas bag, and retorted in the usual way, the time occupied in the retorting being usually seven hours. When custom ores are treated, the arrastra is sampled every half-hour, to ascertain how the working is progressing, as well as to ascertain the real value of the ore. If it is worth while, the tails are afterwards treated in the pans. When the arrastras are used in silver mills, they are discharged, by the plugs in the side, into a wooden trough, which carries the pulp directly into the pans, and the treatment is the same as in a pan mill. The tails are worked in the pans exactly as ordinary tails are treated, the quantity of copper sulphate ranging from 4 lb. to 5 lb. to the charge. The percentage of gold extracted by the first treatment generally reaches 90 per cent.; that of the silver, from 75 to 80 per cent. The arrastra is very often used in gold stamp mills to treat the pulp from the stamps. At the Iowa Mill, in Idaho, and in a considerable number of works in California, the pulp from the stamp flows into a constantly working arrastra, and from there through a gate near the top of the side, on to amalgamated copper aprons and sluices. In such a continuous process, a flask of mercury is charged into the arrastra at the commencement of the run, and is removed as amalgam and free mercury at the clean up.

The amount of labour required in this process is extremely small. One man per shift can easily take care of two arrastras. In some of them which work on tails, and are run by water power, the disposition of the mill is such that the only labour required is in feeding and discharging them, so that it is much less than the work of one man. Very often the owner of the mill does all the work himself. When the arrastras are continuous in action, the amount of labour is still smaller, it being simply that which is necessary for repairs. The power which is required depends upon the size of the mill, but small arrastras are

State or Territory.	Works.	Number of Arrastras.	Number of Drags.	Weight of Drags.	Motive Power.
				lb.	
Alabama	Houston and Pnison, Cleburne Co.	1	6	100 to 300 each	8 horse-power engine; also runs battery.
Arizona	Cross and Smith, Mohave Co.	2	2 each	.. ..	4 horses.
	South Oro Belle, Yavapai Co.	2	4 each	400 each	5 horse-power engine.
	Thunderbolt	4	4 each	175 each	20 horse-power engine.
	Johnson	2	4 each	250 each	2 horses.
	Nash	3	6 each	200 each	1 horse for each arrastra.
California	Leper, Keys, and Co., Calaveras Co.	6	3 each	300 to 500 each	Overshot wheel, 28 ft. dia., 50 in. water.
	Potter, Calaveras Co.	2	3 each	200 to 500 each	Overshot wheel, 20 ft. dia., 25 in. water.
	Champion, Calaveras Co.	2	4 each	750 each	Overshot wheel, 40 ft. dia., 50 in. water.
	Fresno Enterprise, Fresno Co.	3	4 each	800 to 1000 each	
	Italian, Plumas Co.	35	Usually 4 each	500 to 2000 each	Hurdy-gurdy wheels.
Colorado	George Bachelder, Boulder Co.	2	2 each	.. ..	Water power.
	Rock Point, Ouray Co.	4	6 each	200 each	Turbine.
Idaho	Iowa, Boise Co.	1	3 each	250-600 and 800	40 horse-power engine runs battery also.
	Estes, Lemhi Co.	1	.. ..	.. ..	Leffel turbine, 13½ in. dia., 65 ft. fall.
	Norton's, Lemhi Co.	2	4 each	1000-1200 each*	Overshot wheel, 15 ft. dia., 5 ft face, 150 in. water.
	Scales and Wagner's, Owyhee Co.	2	4 each	250 to 300 each	In winter, 30 horse-power engine; other eight months, 20 ft. overshot wheel, 4 ft. face.
	Trask's, Owyhee Co.	2	4 each	300 each	Overshot wheel, 24 ft. dia., 2 ft. face.
Montana	Black Tail, Deer Lodge Co.	2	4 each	2000 to 3000	Water power.
	Glen Egbert, Deer Lodge Co.	1	4 each	500 each	Overshot wheel, 14 ft. dia., 2½ ft. face.
	Schafer, Lewis and Clarke Co.	2	One 4; one 3	500 each	Overshot wheel, 33 ft. dia.
	H. Siegchrist, Madison Co.	2	4 each	800 each	Overshot wheel, 20 ft. dia., using 60 in. water.
Nevada	E. R. Willis, Esmeralda Co.	1	5 each	120 each	Undershot wheel.
	B. B. Chandler, Storey Co.	1	4 each	200 to 500 each	Overshot wheel, 24 ft. dia.
New Mexico	Skillicorn's, Grant Co.	4	4 each	80 each	20 horse-power engine.
N. Carolina	Mann, Nash Co.	10	4 each	200 to 400 each	40 horse-power engine.
Oregon	McCord, Baker Co.	2	4 each	300 to 500 each	Overshot wheel, 24 ft. dia., 12 horse-power.
	Beagle, Grant Co.	1	6 each	300 each	Hurdy-gurdy wheel, 10 horse-power.
	Sugar Pine, Josephine Co.	2	4 each	600 each	Water wheel, 20 horse-power.
	Cooper and Lockwood, Yakima Co.	3	6 each	200 each	Turbine, 20 ft. dia., 13 ft. fall 9 horse-power.
	Shafer, Yakima Co.	1	4 each	500 each	Horizontal hurdy-gurdy.
	Witer and Miller, Yakima Co.	3	5 each	.. ..	Hurdy-gurdy, 8 horse-power, and breast wheel, 20 hor se-power.

* When new; used until worn to 250 lb. or 300 lb.

frequently worked by a single mule or horse. The Table on preceding page gives the number of arrastras, the number and weight of the drags and the method of obtaining the power in thirty-two mills which work 111 arrastras.*

When water power can be had, small overshot or turbine wheels are used, or a hurdy-gurdy if sufficient pressure can be obtained. When the mills are large and have rock breakers or stamps, they are usually placed in such a position that water power can be made available for certain seasons of the year. When, either from drought or cold, the water is not available, steam is used. The simplest form of arrastra, operated by mule power, can be built for from $100 to $150, which is the cheapest form in which it can be constructed. The cost varies from this point up to $1000 or even $1500, depending upon the size of the arrastra and the care taken in making the pavement and the weight of the drags. When flumes, pipes, and ditches are required to bring the water, and water wheels and engines are used, and when pans are added for the treatment of the tails, the works may become very expensive. The complete cost† of twenty-six mills, having ninety-two arrastras, in 1880, was $137,590, or an average of $5292 for each works, or $1495 per arrastra, including all the rest of the plant. These mills, however, are of the best construction, and their cost is considerably above that of the ordinary arrastra mills. The following details refer to the working of Scales and Wagner's arrastra, Owyhee Co.,‡ Idaho, one of the best of all the arrastra mills. It produced in 1880 $205,331.75. The charge for custom ore in lots of 100 tons, and over, is $15, for lots of 50 to 100 tons, $16, and a corresponding increase for smaller lots. The plant consists of a battery of ten light stamps, two arrastras, three half-ton Wheeler pans, two settlers, and a retort.

Details of Treatment in the Summer, on Twelve Hours' Shift.

Four men inside the mill at $4.00 per twelve hours' shift	$16.00
Two Labourers in the summer, on ten hours' shift outside work, at $3.50	$7.00
Two engineers in winter on twelve hours' shift at $5.00	$10.00
Number of days' work in the year	330

* Report of the U.S. Census for 1880, vol. xiii., p. 281.
† *Ibid.*, p. 282. ‡ *Ibid.*, p. 284.

Hours of labour to treat one ton of ore	12.7
Cost of supplies per ton of ore treated	$3.95
Tons of ore treated in 1880 (tails about 300 tons) ...	1,772.25
Lowest yield of ore	$38.00
Highest ,, ,, ...	$600.00
Average	$115.85
Yield of the tails	$9.50

The following Table gives the total amount of supplies consumed, and the amount per ton of ore treated:

Items.	Total Consumption.		Consumption per Ton of Ore Treated.	
	Amount.	Cost.	Amount.	Cost.
		$		$
Red fir . . cords	575	4168 75	0.32	2.35
Quicksilver . lb.	3,000	1440.00	1.69	.81
Salt . . . ,,	14,000	420.00	7.89	.24
Bluestone . ,,	3,000	750 00	1.69	.42
Lard oil . . gallons	20	40.00	0.01	.02
Chemicals and sundries	...	200.00	...	.11
Total . .	...	7018.75	...	3.95

CHAPTER V.

GOLD STAMPING AND AMALGAMATING MACHINERY.

THE decay of shallow placer mining was soon followed by the discovery of auriferous lodes, the ore of which had to be worked by machinery. Quartz mining, as it is called, commenced in California in the year 1851 in Grass Valley, and for thirty-eight years has been carried on there with the greatest energy, persistence, and success. During that year a number of ledges were prospected, a small stamp mill was erected, and several hundred tons of ore worked. In the same year a mill was erected in Jamestown in Tuolumne County. During the following year a much larger and better-equipped mill was erected on the Fremont Estate in Mariposa County. These mills all had wooden stems, and were much less effective than the modern stamp. For the next five or six years gold mining and milling was extensively carried on in different parts of the State, Grass Valley, however, keeping the lead, both in the quantity and quality of the work done. From 1860 to 1875 about 400 stamp mills were constructed, having in the aggregate 4200 stamps, but of this number it is estimated that not over 50 per cent. were in active operation at any one time. Since that date accurate information is not accessible, but it is believed that while the number of mills has not greatly increased the number of stamps and their duty has been materially augmented. Owing to the want of experience in the business and the desire to have something to show, and the consequent expenditure of the capital in surface plant, many of these enterprises proved failures, leaving behind them for many years monuments of what was either folly, inexperience, or downright knavery, in the shape of mill-houses and machinery going to ruin. Some of the mistakes, however, arose from working what in California are known as specimen

mines, or those which, as they yield beautiful samples, are therefore supposed to be very rich, while experience has shown that the most productive mines generally yield no specimens. Others were the construction of mills before the mines justified it, or putting up more stamps than the output of the mine or that of the district, in the case of custom mills, would warrant. Others were mistaking the character of the lode, sinking deep in the hope of finding rich ore further down when the surface ore did not justify it, the want of proper sorting and assaying of the ore, so that poor and rich ores were worked together, the result of which was unprofitable, while the rich might have paid, or owing to want of proper management the rich tails were allowed to flow away, or the use of too much mercury and too much water at too high a velocity, and, more than all, the want of proper business capacity, and the general idea that any one can improvise himself a miner, even without any experience, and thus direct the operations of the mine from long distances. No profession requires greater business capacity, more personal supervision, better mining, mechanical and metallurgical engineers, and better chemists and assayers, than the business of gold mining, and no other profession requires more careful and watchful observation than is required in the daily inspection of vein phenomena. Miners are said to live upon hope and to be unacquainted with the positive degree. Hence any "find" is apt to be exaggerated, and, on account of the uncertainty of the product of the vein, every hope announced as an ascertained truth. Yet while the risks in mining are undoubtedly great, the profits, with sufficient intelligence, are likely also to be great. No other business, managed with equal care and intelligence, on the whole gives greater chances of success than mining. A great many failures have been caused by the neglect of laying up a reserve capital as a provision for a rainy day, and declaring, with a view to affect stock speculations, large dividends with the first moneys earned from the mine. This is not incident to gold mining alone. There are many silver and copper and other mines in this country now leading a precarious existence burdened with debt, kept working only in the hope of better days, which may never and probably will never come, or entirely

ruined, whose crippled condition or complete failure is owing to large dividends which ought to have been invested either in the payment of the debts or in making permanent improvements in the shape of work done ahead, which might have been prosperous to-day had they been managed with proper prudence and foresight. While history repeats itself, all of these mistakes will not be avoided. With educated engineers and metallurgists and experienced superintendents, the conclusion that a poor "prospect" will make a good mine is not so often made, nor the endeavour to start a mine on a small quantity of good ore. The enormous outlays and unwarrantable extravagance of the early quartz mining was only possible because rich free-milling veins were struck which would bear it. It has taken years of unsuccessful enterprises to determine the fact that gold-bearing veins as such are not generally rich, and millions of treasure have either been wasted or lost in demonstrating that they are not always free-milling. The real wealth of the country lies in its poor, and what the miner calls in contradistinction to free-milling, "rebellious" ores. It is the labour, skill, and experience which have been spent upon these ores which has caused the great progress made in treating this problem of so-called "rebellious" ores, and which has produced the great advance in metallurgical and mining science and practice in this country. Quartz veins are not as a rule rich, but they are generally sufficiently reliable to yield to the same skill and intelligence and capital which is used in other industries a fair profit with less risk, but it requires personal supervision and the employment of honest and capable men, who will not work the company instead of working the mine. Prior to 1860 it was thought that veins must yield at least $20 to the ton to be profitably worked, while the average of all the mines now worked will be far below that sum.

Most of the gold ores of California have a quartz gangue, but they sometimes occur in slate. At and near the surface the associated minerals in the vein rocks are the oxidised ores of iron zinc, lead, and copper. At a greater depth these turn into sulphides and arsenides of the same metals. They usually contain from ¾ per cent. to 6 per cent. of pyrites, which generally carries some gold. It has been observed in California that when the

pyrites is crystallised, or when it occurs in the country rock, it rarely carries much gold. Occasionally tellurides are found, as in Sierra and Nevada counties, where they are abundant between the north and south forks of the Yuba River, and also near Carson Hill in Calaveras Co. Native gold is always in a state of alloy with silver. Its specific gravity varies from 13 to 19.25 and its fineness from 600 to 950. Its average value is a little over $16 per ounce.

The ores of Colorado have about the same general characteristics as those of California, but they are much richer in pyrites. In Dakota the gold is found in quartz, slates, and sandstones. The slates are chloritic talcose, and are interlaminated with quartz. These veins vary greatly in size, some of them being 80 ft. and over wide. In other cases they are what is known as "blanket lodes" in sedimentary rocks. They sometimes cross the strata, but are generally more or less parallel to it. Of the crystalline rocks the vein matter is usually the country rock more or less altered, in which certain portions have been replaced by the present vein matter. They usually have but one well-defined wall, and are persistent. In some cases they are simply the impregnations of the country rock with pyrites containing gold which has become more or less decomposed, leaving some of the gold free. In other cases the decomposition of the pyrites is so complete that the tails after amalgamation are not worth treating. In others they are less decomposed and the tails require concentration, while in still other cases the ores require smelting. Each of these classes requires its own method of treatment. Those requiring smelting may be said to be characterised by the presence of manganese, which, while it is oxide on the surface, produces below the water level beautiful crystals of carbonate, and also by the presence of copper carrying both silver and gold, instead of lead, as is usually the case elsewhere. Placer deposits are frequent, which in some cases have been rich enough to warrant carrying the gravel to water, distances of from one-half to three miles.

In North Carolina, brown ores, as they are called, contain most of the gold produced in the State. These are decomposed sulphides, which disappear as soon as the drainage level of the

country is reached. The gold below is contained in sulphurets, generally either of copper or iron, and at some future day will probably be successfully worked when the idea becomes sufficiently extended that many of the non free-milling ores will pay to work.

The gold is generally in small metallic particles, very often not visible to the naked eye; but it is also inclosed in pyrites which, in the treatment, become concentrated in the tailings, and require a separate process for its extraction. Frequently it is so intimately mixed with iron sulphurets that it is difficult of amalgamation, though it sometimes amalgamates notwithstanding the presence of pyrites. At other times the gold is so extremely fine that it is carried off on the water before it can be amalgamated. It is also supposed that a sulphuret of gold is found in some mines. That "coated" gold, or "rusty" gold, as it is sometimes called, *i.e.*, gold which cannot be amalgamated in the ordinary way, exists, is a matter of fact; but there is little authority for the belief that sulphur is ever chemically combined with gold in these ores. Rusty gold is generally coated on the outside with some substance which, until it is removed, prevents the contact of the mercury.

The distribution of the ore in the veins is very variable. Sometimes it is quite uniform through the whole mass of vein rock. At others it is on the hanging or foot wall. Sometimes it is in chimneys or pockets; at others in streaks more or less parallel to the beds or bands of the vein rock or to the walls, or it may not follow any direction, and is distributed with the greatest irregularity. The gold itself is fine or coarse, but follows no law in this respect. Sometimes it can be seen by the naked eye, at others, and generally in the best-paying mines, it is so fine that it can hardly be distinguished with a glass. When it is coarse it is easily caught but the vein is apt to be poor as a whole. Those veins are apt to yield best which show no gold to the eye, and produce no specimens. In most veins pyrites is found and the gold is not only scattered through the vein stuff, but is also intimately associated with the pyrites, so that it will generally pay to dress the tails and treat the concentrates. The amount of gold-bearing pyrites found is very variable, but will

usually be from 1 to 5 or 6 per cent. Sometimes it is strictly found within the walls of the vein, which is separated by a distinct selvage from the country rock. Sometimes it penetrates the walls to such an extent as to make it worth while to remove a part of them, as was the case at Carson Hill, where the slates adjoining the vein were penetrated with gold to the depth of a foot or more, equal in value to the richest part of the vein matter itself. The same is often true of other rocks than slates. But the most favourable condition is that in which the walls and country rock are entirely separated, so that the parting between them is clear and distinct. It is also found extensively in the metamorphic slates which have been at one time impregnated with iron pyrites, the slow decomposition of which has entirely altered the rocks which contain it and left the gold in them free. In fact it occurs in every position in which filtering waters containing gold in solution, or carrying it mechanically, could have deposited it, but as it can only be followed to any extent with certainty in veins, it is usually found and worked from them. Formerly, and even up to 1883, the largest part of the gold product of California came from the gravel mines, but since the injunction, the quartz mines have taken the lead and are likely to retain it until some new method of working the deep placers has been invented.

There are several considerations which must always be kept in view in selecting a site for a gold mill. One, and the first and most important, of these, after insuring the presence of the ore, is that of transportation, which should always, for the ore, be reduced to a minimum both of difficulty and of distance. If anything is to be transported it should be the gold, which can usually alone bear it, as it is but a very small fraction of the ore treated to get it. The quantity of the ore is so large relatively to the output that all transportation of it should be reduced to its smallest amount, and should, if possible, be made automatic, so that from the mine car to the tails it needs no handling. This can always be done by judicious and careful study of the hills of the country, for not only should the ore arrive at the mill from the mine, at the highest point possible, so that large quantities of it may be stored in bins to provide against any accident in

the mine or delay in its shipment, but the elevation of the storage bins above the mill should be such that its movement from that point should be automatic. Each floor of the mill should rest on solid rock, if that is possible; the ground in the hill being excavated far enough back to insure to each of the floors a good foundation. From the mine cars the ore should go to grizzlies, while the coarser material which does not pass goes to the crushers, and from there to the hopper, and so on. The grizzlies are made of iron bars 1 in. wide and from 2 in. to 4 in. deep. They are separated by washers, which are 1½ in. to 2 in. thick. This grating is held together by 1 in. round iron rods. They are built in frames from 3 ft. to 5 ft. wide and from 10 ft. to 15 ft. long, which are set at an angle of about 45 degrees. They last about four years The ore bins at the mill should be as large as the situation of the ground will allow of, so that as much of it can be stored at a time as possible, in order to avoid the delays and inconveniences resulting from a want of ore and the consequent stoppage of the mill in the event of delay in raising it from the mine. They should contain at least one day's supply of ore, and as much more as is possible. The Drumlummon Mill, with 60 stamps, has a bin capacity for 3000 tons. The Treadwell Mill in Alaska, with 240 stamps, has 2000. The Father de Smet in Dakota, with 100 stamps, has 750 tons. The North Star in California, with 30 stamps, 1000. The Ilex, with 40 stamps, 500. The Sickles, with 40 stamps, 240 tons; and the Sierra Buttes, with 60 stamps, 140 tons. The bins should be divided into compartments, so arranged as to discharge into the ore feeder hoppers behind each battery, and to allow of sampling the ores from different parts of the mines by discharging them into one or other of these compartments as desired, unmixed with the general run of the rock. This separation into compartments will be found to be a very desirable arrangement in determining the values of ores in different parts of the mine as the work progresses. The ore bins should be so placed as to discharge directly into the ore feeder hopper, and the angle of inclination of the floor should be such that the slide of the ores will be gradual and constant, whether they be dry and slippery or wet and sticky. The

regularity of the discharge can always be governed by adjustable gates. This requires floors at different levels to provide for the machines to be used. There should also be beyond the mill plenty of tail room. This can all be secured by attention to the topography of the country. It would be possible to set up a mill on level ground, and to obtain the height by high walls of masonry and by timbering, but it is much more convenient and also more economical to secure a side hill for the purpose. Last, but not least, the quantity of fuel and water, and also its cost, should be considered. This involves the question of power, as to whether the work shall be done with water or steam. If water is to be used, the source must be unfailing and its supply constant at all seasons and under all circumstances. Under such conditions only is its use economical, but if the supply is not constant, if it is affected by drought in summer and snow and ice in the winter, it becomes a source of permanent anxiety, annoyance and expense, and should be abandoned altogether or supplemented with steam. The economical use of steam depends on the cost of fuel. In any case it involves a larger outlay for the plant, and its running is besides more expensive, as it involves the expense of boilers, fuel, and an engineer to run them.

Of the ores to be treated, the larger part are low grade and free milling. Usually when they are not free milling they must undergo some more or less complicated treatment, whose object is to get rid of the foreign element. When roasting sets the gold free, the process is simple. Usually, however, some other process than amalgamation will be found most serviceable when the sulphurets are in very large quantity. It has been found, however, in Canada, that arseno-pyrites containing gold can be easily roasted, so that 80 to 85 per cent. of the gold will amalgamate without further treatment. In any case, the ore must be reduced in size before it can be treated. This can be effected in a variety of machines, which have every degree of power, from the rude Mexican rock-breaker, Fig. 1, vol. i., to the most perfect mill of to-day. They may all be included under crushers, stamps and rolls,* which last are only just now being generally introduced.

* Vol. i., p. 210.

The ore is generally crushed in a Blake's crusher* or some modification of it. The Treadwell Mill has six Blakes for 240 tons, the Drumlummon three for 60 stamps. The Bunker Hill Mill has one Hendry's Challenge crusher with jaws 9 in. by 15 in. for 40-stamps. The Gates crusher, Fig. 42, which has for ten years been used in crushing macadam, ballast, and, iron ore, has recently been introduced into the Caledonia Mill and other mills in Dakota, where a No. 6 crusher does the whole work of an 80-stamp mill in ten hours. It is one of the most

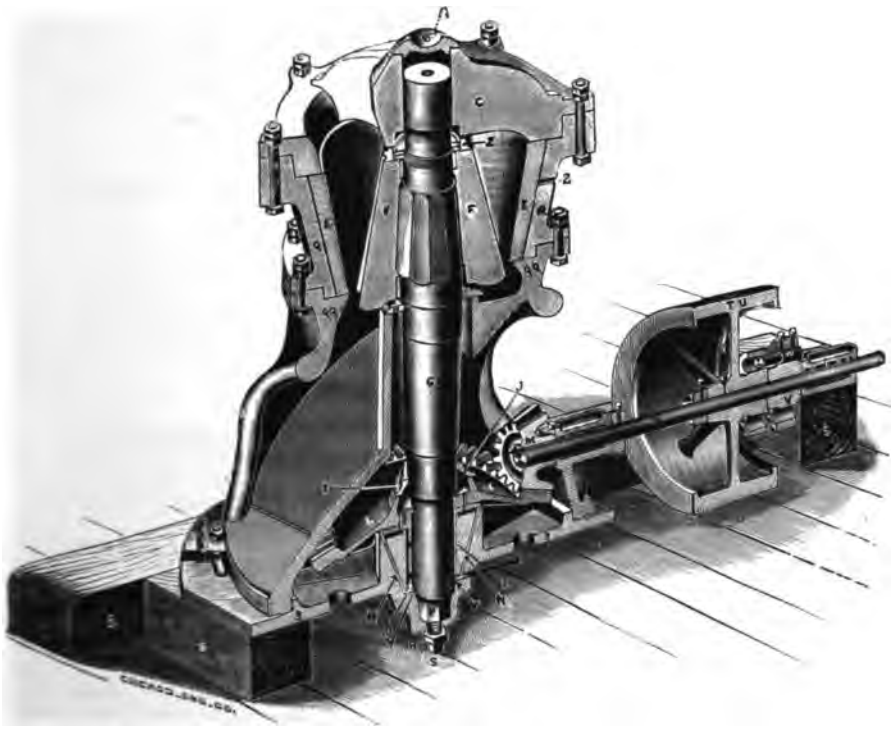


FIG. 42. SECTIONAL PERSPECTIVE VIEW OF GATES' CRUSHER.

A Cap for keeping dirt out of the journal.
C Top of crusher or spider.
D Bronze eccentric journal at the bottom of the shaft G.
E Chilled iron concaves or crushing surfaces.
F Chilled iron head.
G Main shaft of forged steel.
H Collar fitted close around the shaft G to keep dust and fine ore out of the lower part of the crusher.

* Vol. i., page 357.

I Collar fitted close around the shaft G to keep dirt and dust of the journal at eccentric D.
J Oil hole through the collar I, where the journal at eccentric D is oiled.
K Collar cast solid on the hub of the bevel wheel L.
L Bevel wheel.
M Bevel pinion.
N Oil holes, to let oil to the bottom of the journal, at the eccentric D.
O Steel step to the bottom of the shaft G.
P Chilled octagon step under the bottom of the shaft G.
Q The shell.
R Jam nut on the screw S.
S Screw for raising or lowering shaft G.
T Driving pulley.
U Flywheel.
V Break pin hub.
W Break pin.
X Countershaft.
Y Pipe through which the old oil is drawn off and the journal is washed out with hot water at eccentric D.
Z Screw collar to keep the head F from working up.
2 Hole in the shell Q, where the key concave is driven out. This concave is not bevelled on its sides, and it, or one just like it, must be put back in the same place, as a bevelled concave cannot be driven out after once run in with zinc.
3 Cast-iron bottom.
5 and 6 Wood frame to which crusher is fitted.
7 Widest opening where the crushed ore is being discharged, when the head is crushing on the other side.
10 Boiler iron hopper.

effective crushers known. It consists of a vertical shaft of forged steel, which is rotated at the bottom by a bevelled wheel L, placed ½ in. out of centre. On the top of this shaft is a chilled iron head F, and opposite to it chilled iron concaves or liners E, set in the body of the machine. These two form the crushing faces; the distance between them at the bottom can be regulated at will by means of set screws S, provided they do not come in actual contact, which would cause the shaft to break; so that the ore can be uniformly crushed to any size, coarse or fine. The whole shaft revolving around an eccentric at the bottom has a continuous gyratory motion, which crushes without grinding, and is, therefore, well adapted for crushing gold ores. The only limit to its capacity is the ability to get ore into the crusher. When the material is dry the head may, in the smaller sizes, be

Size.	Dimensions of each Receiving Opening, about.	Dimensions of the Three Receiving Openings combined, about.	Weight of Crusher.	Capacity per Hour, in Tons of 2000 lb., passing 2½-in. Ring, according to Character of Rock or Ore.	Dimensions of Driving Pulley.		Space occupied by Crusher.			Diameter of Hopper.	Revolutions of Driving Pulley.	Size† Engine recommended to Drive Crusher, Elevator, and Screen. Indicated Horse-Power.	
					Diameter.	Face.	Height from Bottom Frame to Top Hopper.	Width of Frame.	Length of Frame.			Limestone.	Granite, Ore.
	in.	in.	lb.		in.	in.	in.	in.	in.	in.			
00	2 by 4	2 by 12	500	*——	8	2⅝	24	17	26	13	700	½	½
0	4 ,, 10	4 ,, 30	3,100	2 to 4	16	6	48	30	73	28	500	¼	¼
1	5 ,, 12	5 ,, 36	5,500	4 ,, 8	20	7	54	31	76	37½	475	8	8
2	6 ,, 14	6 ,, 42	7,800	6 ,, 12	24	8	60	39	90	39½	450	12	15
3	7 ,, 15	7 ,, 45	13,500	10 ,, 20	28	10	73	48	103	44½	425	20	30
4	8 ,, 18	8 ,, 54	20,000	15 ,, 30	32	12	85	54	114	51	400	30	40
5	10 ,, 20	10 ,, 60	27,000	25 ,, 40	36	14	93	63	123	59	375	40	50
6	11 ,, 24	11 ,, 72	36,000	30 ,, 60	40	16	109	73	139	66	350	50	60
7	13 ,, 30	13 ,, 90	60,000	40 ,, 75	48	18	116	73	144	121‡	350	60	75
8	18 ,, 45	18 ,, 135	89,000	100 ,, 150	48	20	156	90	164	132‡	350	125	150

* The size 00 is intended for laboratory and sampling purposes. When running at full speed, it will break 250 lb. per hour of any rock or ore down to ⅛ in., or 500 lb. to ⅜ in. It has a wheel to run by hand.

† Power.—In estimating power required to drive these crushers, ample allowance is made to provide for running an elevator and screen also. But it must be borne in mind that no close estimate can be made to cover all sorts of rock and ore; and further, it should be observed that it requires much more power per ton to break rock to ½ in. than is required to break it to 1 in. The above tabulated estimates are intended to cover the ordinary crushing. For breaking the hardest rock to 2½ in. one horse-power per ton of rock broken per hour is required.

‡ Very heavy cast-iron hopper. All other sizes have boiler iron hoppers.

When sizes 0, 1, 2, and 3 are set fine as practical the product will vary from ¾ in. down to sand; sizes 4, 5, 6, and 7 from 1¼ in. down; size 8, 2 in. down, if the ore is clean.

set to $\frac{3}{8}$ in. from the concaves, allowing only this size to pass. If set any closer the crusher would be liable to pack and the shaft to break. Every arrangement is made for oiling the machine and for taking it apart. The repairs, with judicious use, amount to almost nothing, as the set of concaves E will last more than two years. When worn out they can be replaced by driving out the key concave, which has flat sides, while the others are bevelled, with an iron bar through the hole 2. The flat-sided key concave must always be placed over the hole, otherwise the other plates could not be removed. The only danger in the machine is setting the head F too close to the concaves E. At the Caledonia Mill, where it was placed in the most disadvantageous position, the only objection made to this crusher was that it did too much work, the whole work of the 80-stamp mill being done by two men in ten hours, and that consequently it had to be done in one place, which, without a special arrangement of shoots to the ore bins, might be in some cases an inconvenience.

These crushers should be placed below the coarse ore bin and directly in front of it. The fine ore which goes through the grizzlies should not be allowed to enter the jaws of the crusher, but should go directly to the self-feeder bin. A shute should carry the coarse ore direct to the crusher, the supply being regulated by a gate worked by a rack and pinion. The opening of this gate can be so regulated as to make the work constant, and in such crushers as the Gates no care is required at the crusher except such occasional oversight as may be easily given by the men on the floor below. From the rock breaker the ore should discharge into the feeder bin without labour.

A number of small mills have been invented for crushing, a few of which are worthy of notice. The Huntington Mill, Fig. 43, is constructed on the principle of combining the action of both the Stamp and the Arrastra. It consists of an iron pan, into the top of which a ring, B, is set, in which are three stems, D, with free motion on the top, to which steel shoes, E, are fastened. These shoes, or rollers, as they are called, and their stems, swing at the top, and

revolve around the pan by means of the central shaft, G. They act on the ore by centrifugal force, the power of which depends on the number of revolutions of the mill. In front of each roller is a scraper, F, which keeps the ore from packing. The bottom of the pan on the outside is surrounded by a deep gutter, on to which the screens open, and from which there is a passage leading to the copper plate sluices. On the bottom of the pan there is a repair piece, and against the sides, where the greatest amount of wear is, there is a replaceable ring. The rollers, E, come down to

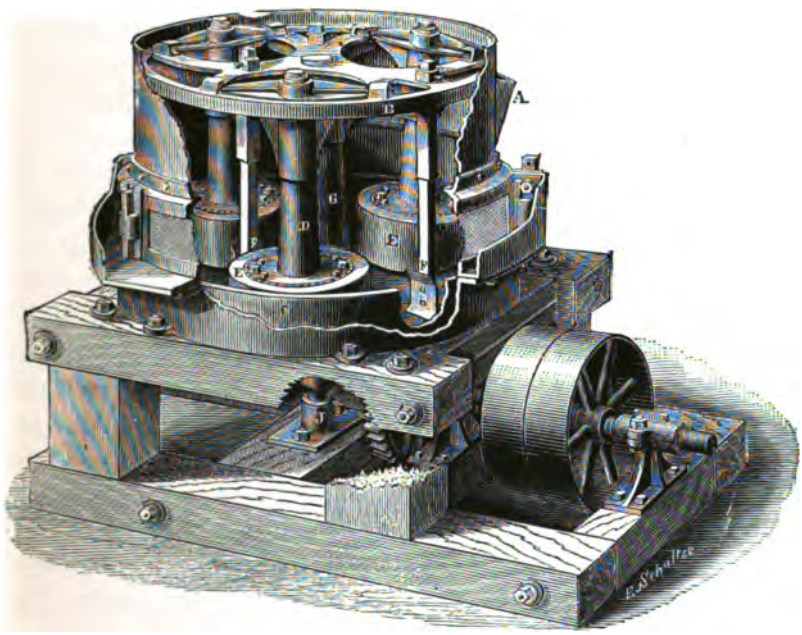


FIG. 43. THE HUNTINGTON MILL.

about 1 in. from the bottom of the pan, but never touch it. The crushed ore and water are delivered into the tub by a hopper, A, on one side, generally by an automatic feeder. It is at once thrown against the sides by the scrapers, F, and the rollers, E, where it may be crushed to any desired fineness, which is regulated by the screens. The splash of the shoes forces the pulp through the screens to the outside trough. The motion of the shoes, which is a swinging one as well as a movement of

rotation in two directions, gives a grinding and crushing action at the same time, while the scrapers, F, prevent the material from accumulating in any one part. The roller first crushes the rock, then by its oscillating motion drives it along a little, acting at the same time like an arrastra as well as a stamp. As the rollers or shoes are suspended from the top above the muller, they pass freely over the mercury and amalgam without grinding it, but the motion is sufficient to agitate the pulp in such a way as to make every portion of it come into contact with the mercury. The pan is built in three sizes, the smallest of which is made in sections for packing it on mules where transportation is difficult. As the splash is heavy against the sides, the abrasion on the screens is very great, but they do not wear as rapidly as would be expected, and are so arranged that they can be taken out and replaced with but little difficulty. From the position of the rollers their rotation in the pan does not flour the mercury by grinding it, while the motion is such as to force the pulp into contact with the quicksilver and discharge it as soon as it is fine enough to pass the screens. The annular discharge around the bottom of the pan is so large that the work of the mill is a maximum, and as there is an opportunity for the pulp to be discharged as soon as it is fine enough, but very little slimes are made. The material, therefore, passes out of the mill into the sluices in excellent condition for concentration. It is, however, only adapted to wet crushing. It runs with considerably less power than the California stamp, and is applicable to the treatment of clay ores which do not work well in the stamp. The cost, also, of the mill is much less than that of the ordinary California stamp.

The following Table gives the details of this mill:

Size. Diameter.	Weight.	Number of Revolutions.	Capacity.	Horse-Power Required.
ft.	lb.		tons.	
3½	5,300	90	12	4
5	11,000	70	20	6
6	20,000	55	30	8

About 150 of these mills have been set up, mostly in the United States. They are largely used in small mines or those worked with but little capital, there being from one to five of them in each mill. A mill with twenty-two 5-ft. mills and one with nine 3½-ft. mills have been erected in New South Wales. Their advantages are, small first cost, as well as expense for transportation to and erection at the mine; the short time it takes to erect them and get them into working order; their action both as crusher and amalgamator, with but small loss of mercury; the fact that they require but a small amount of power, all of which is efficiently used, and that the working parts are easily replaced. It is also claimed for it that it saves the floured gold, which in most mills is lost.

The Allenton Mill, near Dayton, Nevada, runs on quartz rock, has two Huntington mills, which are fed by a Huntington feeder, which has a backward and forward movement but discharges the rock very regularly. These mills are 3½ ft. in diameter and have double bottoms and steel liners. The bottom weighs 300 lb. and is of cast steel. The rings are of steel and weigh 500 lb. The shoes are of cast steel and weigh 100 lb. They last twenty days. The bottoms wear four months. The rings wear forty days. Each mill crushes 10 tons a day. The screens are No. 4 slots. They last only four to five days. Mercury is added in the mill once a day. The rings cost $30, the bottoms $12, and the shoes 12 cents a pound.

At the Spanish Mine, Nevada County, California, eight miles from North Bloomfield, the ore worked is a decomposed slate containing stringers of quartz, with clay partings between the slate and the quartz. The most of the gold is found free in the clay, some in the quartz and but little in the slate. The ore is very poor, yielding on an average only 60 cents a ton. The method of mining consists in running in adits, and from there an incline to the surface, and then stoping down the ore, or mining it from an open cut. The ore is first crushed in a Blake's crusher and discharged into a bin and from there on to a challenge feeder which is worked by a shaft, on which are carriers, which strike the feeder at regular intervals. It falls into four Huntington mills, three of which are 5 ft. in diameter and one 4 ft., the

bottom and sides of which are protected by liners. The mills revolve sixty times a minute. The shoe weighs 125 lb. and is made of rolled steel. Seven a month are used in the four mills. When they are too much worn to be used in the large they are taken out and used up in the smaller mill. There are three shoes and scrapers in each battery. The screens are slot screens, No. 5, and are set in frames 9 in. by 31 in. No. 6 screens were used for a long time, but No. 5 has been found to answer better. The screens last four days only. One new screen a day is used for the four mills. The cost of the screens is the most expensive repair in the mill. The large mills grind 35 tons to 40 tons of rock a day; the small ones 30 tons. As the force of the shoe is against the sides of the pan, the splash against the screens is very great, much greater than it could be in an ordinary battery. The false bottom and liners last a year. The discharge from the crusher is circular, and is conducted from the sides by means of a trough around the pan to a discharge, so that sluices could not be conveniently put in in a straight line.

As the space in these works was extremely contracted, the copper plates were made in layers of four, one over the other. To gain room and to prevent their being scoured, the pulp is made to fall on to an iron plate with holes ½ in. in diameter, fitting over the amalgamated plates below. It was found that the attempt, at great inconvenience, to place the plates in line simply caused the upper end ones to scour. The top plate is 4 ft. long by 2 ft. wide. The next one is 3 ft. long by 4 ft. wide. The next 4 ft. long and 4 ft. wide. The lower one is 4 ft. by 4 ft. This arrangement of plates is repeated twice. From the last plate the pulp runs to waste. These plates cost $4 the square foot. The plates are dressed every morning. The clean-up is made once a month. Mercury is added in the battery at the rate of 1 oz. to 4 tons of ore. There are no amalgamated plates in the pan. The loss in mercury is 1 oz. for every 15 tons to 30 tons of ore crushed, the exact amount depending upon its richness. The mill is run by an 8-ft. Pelton wheel with 15 in. of water at a pressure of 320 ft. Twenty-two horse-power is required to run the mill; 950 oz. of mercury a month are used in the four mills,

the total loss being 160 oz. The entire plant cost $14,696, including the installation of the water power.*

	$
Four Huntington mills and self feeders...	6,346.22
Labour, setting up and building	2,304.83
Silver plated amalgamating plates	1,985.35
Water pipe and wheel, shafting and pulleys ...	1,220.75
Lumber, building, and V flume	1,194.52
Hardware	1,026.08
Blake crusher	618.45
Cost of milling plant under cover, and running ...	$14,696.20

Freight $24 per ton from San Francisco; lumber, $22.50 per thousand.

The four mills crush from 120 to 140 tons per day, depending upon the proportion of quartz in the ore. Amalgamation is done in the mills, 45 per cent. of the gold is saved around and inside the mills, and 55 per cent. on the plates. The tailings run to waste after leaving the plates; very little of the gold or amalgam is floured.

The following Tables give the cost of treatment by the Huntington Mill at the Spanish Mine. During the last four months of 1886 the mills crushed 136 tons a day. The average expenses for that time are given below:

	Per Ton. $
Labour { Tending mills—cleaning up / Feeding mills—superintendence } ...	.09
Water (for rock breaker also)...	.036
Shoes	.029
Screens	.013
Dies	.017
Caps, scrapers, and bolts	.002
Renewal of working parts	.02
Quicksilver (at $40 per flask)...	.005
Oil for illumination and lubrication including the rock breaker	.002
Depreciation	.01
Cost of milling per ton. 22 cents four mills... ...	.224
Labour at rock breaker	.02
Wear and tear of rock breaker	.005
Total cost per ton, 24 cents nine mills	.249

The item of 2 cents for renewal of working parts is too high, but it provides for all possible contingencies.

* For this information I am indebted to the superintendent of the mine.

The cost for the month of November, 1887,* computed on the treatment of over 4000 tons, which yielded $0.65, was:

MINE.

Ore produced in Thirty Working Days, 4047 Tons.

Cost of Production.	Labour.	Supplies.	Total.
	$	$	$
Extracting ore . .	679.63	196.65	876.28
Delivering ore to mill .	193.25	13.69	206.94
Dead work . . .	100.90	14.35	115.25
General expense . .	70.70	4.75	75.45
Total . . .	1044.48	229.44	1273.92
Cost per ton . .	25.8-10	5.6-10	31.4-10

MILL.

Ore reduced in Twenty-Nine Working Days, 4047 Tons.

Cost of Reduction.	Labour.	Supplies.	Total.
	$	$	$
Mill expense . .	225.67	162.82	388.49
Water for power . .	5.00	198.00	203.00
Handling ore . .	177.00	2.40	179.40
General expense . .	70.71	4.75	75.46
Total . . .	478.38	367 97	846.35
Cost per ton . .	11.8-10	0.09	20.8-10
Bullion produced .	...	...	2644 57
Total expenses . .	...	...	2120.27
Profit . . .	...	...	524.30

The cost of mining and milling was about 0.52, making for November, 1887, a profit of $0.13 per ton. This statement was made under oath to the preferred creditors of the mine, and is probably the best result ever obtained from low grade ores. In September, 1888, the mine was yielding only $0.60 and was still working with a profit.

At Angles Mill, at Angles, Calaveras County, California, the milling is done by a roller crusher machine, invented by J. B. Lowe, of San Francisco. This consists of a circular iron cylinder, revolved by gearing and running on friction wheels. This cylinder is 4 ft. long and 5 ft. in diameter. In the centre,

* For this information I am indebted to the superintendent of the mine.

separated by an interval of about 9 in. are two cast-iron balls, 14 in. in diameter, and weighing about 1200 lb., which revolve on the bottom of the cylinder, which makes 32 to 34 revolutions a minute. On either side of the revolving compartment, holding the balls, is another, 8 in. wide, with partitions arranged like an overshot water-wheel. The ore is fed through the centre near the axis of the revolving drum by an automatic feeder. The spout projects to near the centre of the cylinder. Below this opening screens are placed, made of No. 10 slotted iron. The centre of the drum only is movable; the outside is stationary, and is covered with a sheet-iron cylinder, 18 in. wide, on the bottom of which are amalgamated plates. These plates are 16 in. wide, 3 ft. 6 in. long, and are protected from the ore falling out of the revolving drum by a screen filled with $\frac{1}{8}$ in. round holes, which prevents the scouring of the plates, and also distributes the amalgam. On the back end of this fixed drum the plates incline up 18 in. Each of these machines is capable of crushing from 14 tons to 18 tons in twenty-four hours. The rock is quite soft. It requires 10 horse-power to run the mill, and $\frac{7}{8}$ths of the gold is caught on the plates and screens outside of the cylinder. A few ounces of mercury are put into the cylinders. From these cylinders the pulp runs on to aprons, which are 3 ft. 6 in. wide and 4 ft. long, with an inclination of $1\frac{3}{8}$ in. to the foot; then with a drop 5 in. high into launders 12 in. wide and 4 ft. long, and from here on to six Frues, two for each cylinder. The ore runs from $5 to $6, with not more than 2 per cent. of sulphurets, which run as high as $80. The bullion is worth $16 to $17 the ounce. The machine has worked well and given good results, but it is composed of so many pieces that some one of them is constantly getting out of order, and the stoppages for repairs are very frequent and cannot be foreseen. The output is not very much larger than an ordinary stamp, but the difficulties of repair are very much greater. A sample taken from the sluices gave the following results when screened through the number of screens given below:

No. 30	Screen	... ...	$15\frac{1}{2}$ oz.
,, 40	,,	... ...	$\frac{3}{4}$,,
,, 50	,,	... ...	$\frac{3}{4}$,,
,, 60	,,	... ...	$5\frac{1}{4}$,,

No. 80 screen	...	...	...	...	...	...	2¼ oz.
,, 150 ,,	...	...	...	...	...	...	13 grains.
,, 200 ,,	...	...	...	...	...	...	58 ,,
,, 200 ,, (through)	...	...	...	...	...	52 ,,

Another mill which has acquired some celebrity is the Tustin Mill, which is a cylinder 54 in. in diameter and 18 in. wide. Slotted dies are used on the circumference of the wheel and wire screens outside of them. Inside of the cylinder, rolling upon the dies, are two cast-iron rolls, weighing 700 lb. and 1000 lb. each. The cylinder revolves by a gear-wheel twenty-five times a minute. The ore is fed by a suspended hopper through one of the journals, and when it is crushed fine enough drops through the screens; 30 tons of roasted ore, and 10 to 20 tons of raw ore, depending upon its hardness, can be put through the mill with 3½ horse-power per day. It has been used to some extent, and takes but little room for so large an output, but does not wear very evenly. This mill is much larger than — though very similar to—the ball pulveriser* so extensively used twenty years ago. The following are some of the details of the use of this machine at the Willard Mill, Calaveras County, California.†

Number of Tustin mills	4
Revolutions of Tustin mill per minute...	20
Duty of Tustin in twenty-four hours, tons crushed	10–11
Size of screen in Tustin, mesh	20
Miners' inches of water used in twenty-four hours in Tustin mills	100
Miners' inches of water used in twenty-four hours in mine...	130
Pressure of water in Tustin mill, in feet	160

Other mills resembling the Edge or Chilian Mills, with battery sluices attached, are also used at small mines.

The California stamp mill still holds its place in crushing against all competitors, but on many accounts it is not a rational machine, and will undoubtedly be eventually superseded. What is needed in crushing a gold ore is to break the gangue up in such a way that both the fine and the coarse particles of gold as well, shall be found in the crushed material in exactly the same condition that they were in the rock, without any change of form, or, in other words, that the ore should be cracked and not crushed. This cannot be done in the stamp, and is less likely to take place

* Vol. i., p. 129.

† Report of the State Mineralogist of California for 1887, part 2, p. 42.

when the ore is associated with arseno-pyrite, which is very brittle, but could be done by judiciously arranging rolls. One main source of loss in the treatment of gold is the change of form which it undergoes in stamping, and this is entirely independent of chemical loss. Some recent experiments made in California have shown that certain ores which yielded only 40 per cent. of their gold contents with the stamp yielded over 80 per cent. at once in machines whose object was disintegration or cracking rather than crushing. Large steam stamps like those previously described* are being built in the West. The Leavitt† stamp has been introduced into the Highland Mill in Dakota.

The older method of doing the work was to crush coarse in the battery and grind in the pans. The more modern is to crush fine in the battery and concentrate. Grinding in the pans involves the use of greater power, greater wear of the machinery, and greater loss in quicksilver, and does not always give better results. The more recent method of crushing fine in the battery, keeping the shoes barely off the dies, is the better of the two. It seems incredible that by this process more gold may sometimes be extracted than by the older method. All ores will naturally break at the point where there is most mineral contained in the rock and form a fine pulp in the battery, when the mineral will be almost entirely disengaged from it. Quicksilver always catches gold and silver when it is free, but in grinding in the pan it is forced into contact with the base metals, which renders it more or less inactive for the precious metals. The result of this process therefore is that more bullion is produced, but the bullion is of less value. Attention is being called to the fact that there are cases where amalgamation without grinding gives as good, if not better, results than the older and more expensive method of crushing coarse in the battery and grinding in pans.

Most of the mines of California are characterised by a small proportion of sulphurets, which usually vary from 1 to 6 per cent., which are moderately rich in gold, varying in yield from $25 to $150 to the ton of concentrates, and in a few cases as high as $1500. But in Colorado the quantity of sulphurets is much higher and

* Vol. i., p. 193.

† Trans. Am. Soc. Mechanical Engineers, vol. vi., p. 377.

their yield lower. The method of working will consequently be different. The California mill may be said to be characterised by shallow mortar, rapid motion, short drop, and coarse screen. The Colorado mill, on the contrary, has a deep mortar, slow motion, long drop, and fine screen, the object of which is to keep the ore in the battery the longest possible time, so that the conditions necessary to the best crushing are sacrificed to that of the best amalgamation, and the stamp mill becomes a sort of amalgamating machine. While the California mill works with a 40 screen, the Colorado mill works with an 80. Pulp crushed through a 6-slot screen in California is found to be made up as follows:* No. 1, 19 per cent. of slimes which remain suspended three minutes in still water; No. 2, 51 per cent. of slimes which pass 640 holes to the inch, No. 1 not being counted; No. 3, 23 per cent. of slimes which did not pass 1600 to the inch, without counting Nos. 1 and 2; No. 4, 7 per cent. which pass 1600.

A similar experiment at the Bobtail Mill in Central City, Colorado, gave the following result:

	Per Cent.
Caught on a 40-mesh screen	1.17
,, ,, 60 ,, ,, ...	17.55
,, ,, 80 ,, ,, ...	13.08
,, ,, 100 ,, ,, ...	10.33
Passed a 100 ,, ,, ...	57.87

It is thus seen that the question for each mill to consider is not how much it can crush, but what degree of fineness is best adapted to each ore, and that the point of too fine ore or floured mercury is easily reached; besides which, in very fine stamping, when a large quantity of pyrites is present, its decomposition renders the water slightly acid, which coats or prevents the contact of the mercury, and thus occasions loss. Yet the coarse crushing of California could not be used in Colorado, nor the fine crushing of Colorado in California.

The following Table gives the results of a series of experiments made during the summer of 1888 on mills in widely different localities and on various kinds of ore. In almost every instance the ore passing the 150 screen was richer than the original ore, and in some cases it assayed five times as high.

* Trans. Am. Inst. Min. Eng., vol. xi., p. 34.

EXPERIMENTS IN SCREENING.

Name of the Mill.	Locality.	No. of Screens in the Battery.	Percentage of Ore left on the Screen.								Percentage passed through 200.
			20	30	40	60	80	100	150	200	
California . . .	Nevada	2 and 3 slot	6.59	23.64	18.34	5.00	9.56	15.12	2.41	11.51	7.81
,, . . .	,,	2 ,, 3 ,,	5.76	24.25	10.92	18.00	17.09	3.83	4.57	8.93	6.83
Bunker Hill. . .	,,	8 ,,	Trace	6.82	10.76	15.59	17.97	6.72	13.99	14.35	13.79
Zeile	,,	4 ,,	,,	6.91	5.35	10.94	9.60	6.97	14.04	6.59	39.57
Spanish* . . .	,,	5 ,,	,,	13.32	6.23	11.58	9.97	10.07	11.07	12.17	25.59
Keystone . . .	,,	8 ,,	,,	3.80	8.16	28.32	26.25	14.99	11.94	5.26	1.28
Drumlummond .	Montana	30 wire	,,	Trace	10.00	15.00	35.00	...	25.00	...	15.00
Jay Gould . . .	,,	4 slot	,,	,,	52.00	8.00	18.00	...	3.50	...	2.5
Golden Star. . .	Dakota	6 ,,	,,	,,	3.59	9.28	12.44	11.78	21.98	26.46	13.46

* Huntington Mill.

Hand feeding of the stamps has been of late almost entirely abandoned. Self-feeders are used in almost all the recently constructed mills. Their introduction has increased the output of the battery as much as 20 to 25 per cent., besides effecting a considerable increase in the life of the shoes, dies, and screens. Of these feeders the "Stanford's feeder"* was the first used. This was followed by the "belt carrier feeder," invented by T. A. Cochrane in 1873, which was introduced into many of the mills. This was followed by the "roller feeder," Fig. 45, patented in the same year, which was made almost entirely of iron, the discharge being regulated by a hollow cast-iron roll. A few months later in the same year the "Tullock feeder," Fig. 46, was invented. This was followed by the "Challenge feeder," Fig. 47, patented in 1874, more than a thousand of which have been manufactured. In 1875 Stanford patented an improvement on his feeder, Fig. 44, which was much superior to his former one. In 1876 the "revolving barrel feeder," and in 1877 the "boss feeder," were invented. The Loftus feeder, a complicated machine, is in use at the Ilex Mill. Four of these only, however, are in general use, the Stanford improved, Fig. 44, the roller, Fig. 45, the Tullock, Fig. 46, and the Challenge, Fig. 47. Both the Challenge and the Tullock feeders can be used either on wet or dry ores. The Stanford and the Roller feeder are not adapted to both of them. Though it was demonstrated from the first that the use of most of these feeders are very advantageous and a great saving in labour, it has taken twenty years to introduce them into mills generally, owing to the opposition of both mill-owners and men. The principle on which most of those in general use are constructed is to carry the ore by a uniform motion up to the point where it falls off the feeder apron into the battery, and discharges it there regularly. Most of the earlier feeders were constructed on the principle of discharging the ore by a jerk, the motion being given by a direct blow of the tappet of the stamp, which may answer more or less well for perfectly dry ores, but does not answer at all for wet or sticky ores. The only successful battery feeding is that in which the ore is kept in the mortar in such quantities only that iron will

* Vol. i., p. 181.

almost touch iron, but the moment the quantity becomes greater the stamp cushions on the ore, the drop is reduced in height, and when the quantity becomes very great a large percentage of the work of the fall of the stamp is lost and the screens are in danger.

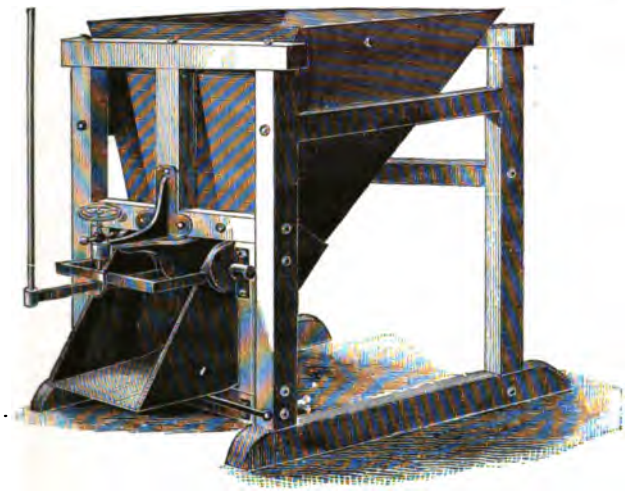


FIG. 44. STANFORD'S IMPROVED FEEDER.

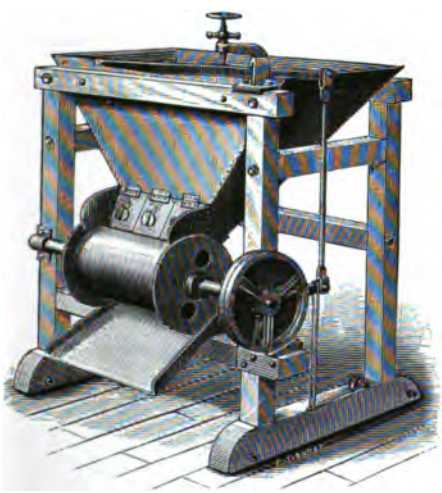


FIG. 45. THE ROLLER ORE FEEDER.

This was one of the principal causes of opposition to the feeders, the first of which, as the ore was discharged by a jerk, often discharged more ore at a time than they should do. To be most effective, the rod of the lever bar should be so adjusted that the

stroke of the tappet shall cause it to feed low, when the maximum amount of work will be done. Stanford's improved feeder was invented for dry ores, but the irregularity of the discharge and the fact of its not being adapted to wet ores led to the invention of the Belt feeder, which was subsequently replaced by the roller, or Templeton feeder, Fig. 45, which had all the advantages of the Belt feeder and was superior to it in the fact that all the joints were made of iron. The jerk is for the most part done away with by the spring attachment, and when the ore is not of the same size it discharges unevenly. Too large a piece may stop the discharge, a flat piece may slide through or clog it, and the roller may even move for some time without discharging any ore. Too fine ore goes through with a rush and fills the mortar too full, and if the feeder is set coarse the fine ore slips through too fast, if set fine the coarse ore does not pass and is apt to stop it. The two feeders in most general use, however, are the Tullock and the Challenge. The Tullock feeder, Fig. 46, is used in the Deadwood Terra Mill in Dakota, and in the California Mill,

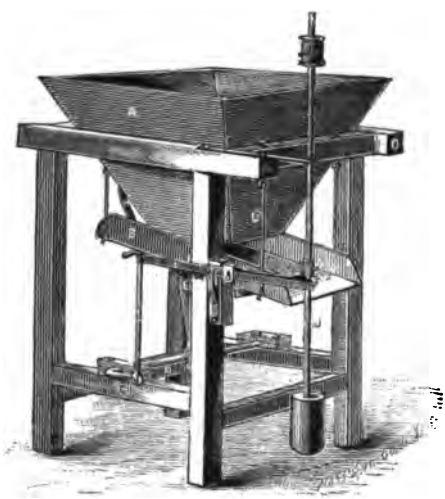


FIG. 46. TULLOCH'S AUTOMATIC ORE FEEDER.

Virginia City, Nevada, consists of a stout frame, 4 ft. 6 in. high and 3 ft. 6 in. square, into which a hopper A, which holds 1500 lb. of ore, is placed. Beneath the hopper is a tray B, set at any desired angle and suspended from the frame by the rod

C, so that it has a movement backwards and forwards inside the frame. Beneath the tray is a rocking shaft D with its crank G, an arm of which, E, is secured by a pin to lugs under the bottom of the tray. The crank G connects with the lever arm I, by which the movement of the tray is given. The arm is attached to the frame near its centre. To the upright stem J an adjustable collar is affixed, so that when it falls it will strike the lever arm, and by the motion of the rocking shaft will throw the tray back. When the rod is raised the tray will move forward by gravity until it is stopped by the lugs striking the bar, which limits its movement. A spring M is sometimes used to aid this movement, but is not necessary. The back of the hopper has an adjustable scraper, and at each motion of the tray a determined portion of the ore is scraped forward by the shock and made to fall into the battery. The hopper is filled from the ore bins, and the quantity discharged at each stroke can be easily regulated to a very small quantity. The motion of the rod is given by the descending motion of the stamp, so that the work is automatic and the supply perfectly regular. This feeder is a cheap one and can be repaired by any blacksmith.

The Challenge feeder, Fig. 47, is constructed so that the tray beneath the hopper has a rotary motion in a horizontal plane

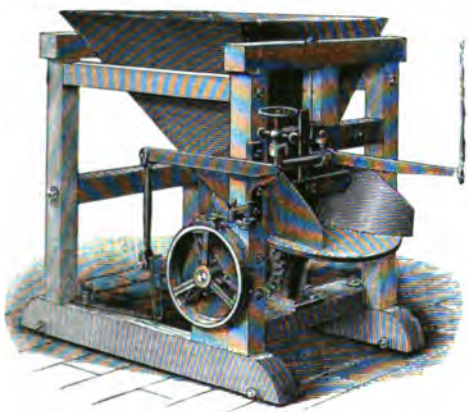


Fig. 47. The Challenge Ore Feeder.

given to it by means of gear wheels, which are moved by a friction grip placed on the outside of the frame, so as to vary

the motion with the blow of the tappet. At each partial rotation a given quantity is scraped off by movable wings, so that the supply is perfectly regular. The bumper rod is placed in the middle of the frame so that there is no necessity of making them right and left. It weighs with the frame about 900 lb. It is specially adapted to very wet ores. It is usually constructed to stand in a frame, as in Fig. 47, or when that is not convenient it may be suspended from a beam. It is the most expensive of all the feeders, but it is the one in most general use. Fig. 48 shows this feeder attached to the battery.

The early feeders were all made right and left-handed, which was a great inconvenience; but by making the bumper rod rise in the middle of the front of the hopper, so as to be acted on by the third stamp of a five-stamp battery, this is no longer necessary. One feeder is required for each battery of five stamps. They take but little power and need but few repairs. They are simple, compact, and will last as long generally as the mill. They are movable on a truck and in no way hinder the work of the mill, and all of their parts are accessible from all sides. They feed easily, regularly, and quietly. Since the feed is so regular they save 25 per cent. of the wear of the shoes and dies and 50 per cent. of the wear of the screens. By an actual test in California, it has been found that $2\frac{1}{2}$ tons per battery of five stamps, more ore is crushed in twenty-four hours when they are used than when the battery is fed by hand. In some silver mills they have increased the quantity 5 tons per battery in twenty-four hours. As the feed is so regular more gold is caught in the battery, owing to the steady and even flow of the pulp over the plates, because the danger of scouring is so much lessened. As the bulk of the gold is caught in the battery, this should have been an anticipated result. With the modern practice of treating large quantities of low-grade ores, it would hardly have been possible to work profitably but for these automatic feeders.

The feeders are always set back of the battery, one feeder being used for five stamps. The simplest arrangement is to have the bumper rod in the middle of the frame acted on by the middle stamp, so as to avoid right and left-handed machines. The adjustment must be made so as to give a constant stream of ore.

The best results are obtained when the feed is low, so that iron almost touches iron ; but with such an arrangement, if the stream of ore should fail, even for a very short time, iron would come in

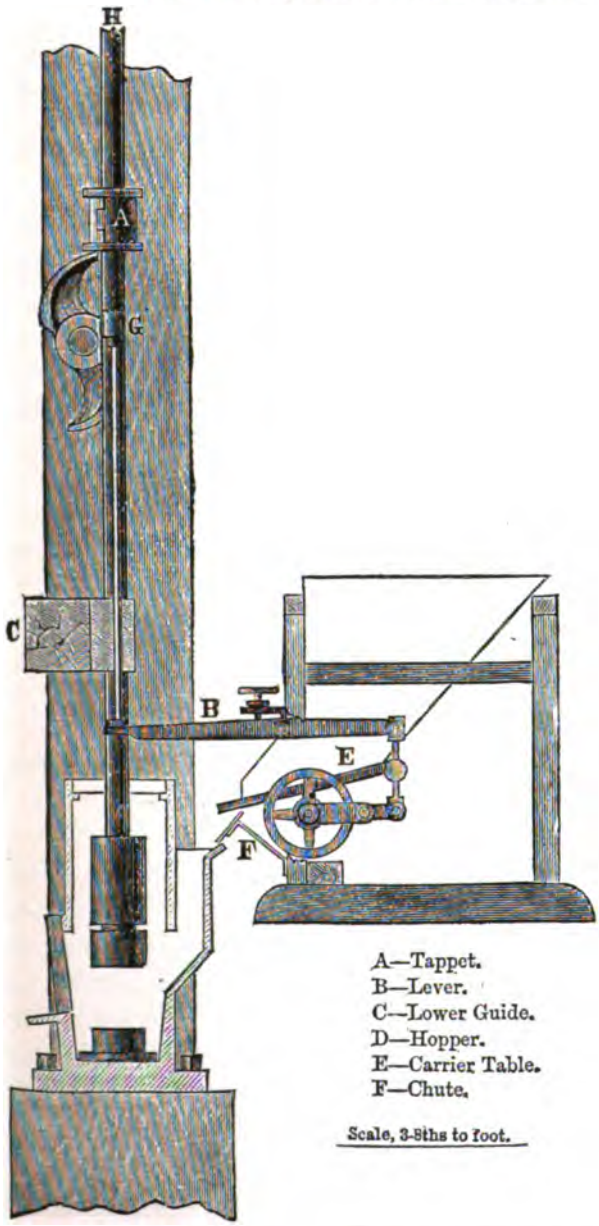


FIG. 48. CHALLENGE FEEDER ATTACHED TO BATTERY.

contact with iron, and there would be danger that the stamp would break.

The ore from the mines usually undergoes a rough sorting, in which, as far as possible, the rock without value is rejected in the mine, and again on the outside. It then generally passes over grizzlies to a crusher, the jaws of which are set from 1 in. to 2 in. apart, and passes through this into a hopper, from which it is discharged into the batteries, either by hand or by automatic feeders. In some cases neither grizzlies nor crusher is used, though the use of the crusher increases the duty of the battery nearly 20 per cent. In either of the methods the amalgamation may or may not be begun in the battery. In the former case, the gold is collected partly by free mercury, which catches it in the bottom of the mortar, and partly by amalgamated plates, on the sides, and especially on the sills and apron. What passes through the battery passes over amalgamated copper plates to two or more sets of concentrators. When battery amalgamation is not used only that gold which is too coarse to pass the screens is caught in the battery, and of this there is a considerable quantity in the ores of Grass Valley. What passes the battery goes over blanket sluices, and what passes them, to the Eureka rubbers and copper plates. What is caught on the blankets also goes through the Eureka rubbers and copper plates. The concentrators which are now used for the treatment are generally the Frue vanner, which has been most successfully used, the Triumph, and Hendy's concentrator, which latter is a much older machine, and is now almost entirely abandoned.

There is a great variety of opinions with regard to battery amalgamation. Some who have used it for years maintain that it is the most economical, while others are as resolute in declaring that it is not so. Its use characterises whole districts, and when it has been for a long time in successful operation it is undoubtedly the best possible, whatever may be the objections to it. As a result of the series of experiments made at the Keystone Mill, in Amador County, and the Eureka Mill, in Plumas County, to test the relative efficiency of the blankets and amalgamators, it was found that the material from the concentrators yielded more gold than that from the blankets, and as a consequence of these

experiments, concentrators were adopted, and the blankets were thrown out. At the Idaho Mill, in Grass Valley, blanket washing is still used, although it is more expensive in labour and the tails are very rich.

In battery amalgamation the mortars are provided with a splash plate outside of the battery, on which amalgamated plates are fastened. The screens are not usually so fine as in silver stamping, for it is found that when the speed of stamps is over seventy, the loss of the gold on the blankets is much greater than with fifty. The object of the stamp mill should be to crack the ore and loosen the gold, and not to flatten it. While it has been found impossible practically to arrive at such a result, every effort is made to come as near to it as possible. The speed with which stamps are run varies from sixty to ninety drops. The more usual speed is about sixty.

In the arrangements* of a gold stamp mill there must be secured, first, constant supervision of the batteries, tables, and sluices, and consequent safeguard from robbery; second, free access to all parts of the machinery, and room for handling it; third, very large ore bin capacity to insure steady milling; fourth, the least possible manipulation of material during the time of such reduction. The gold, like the silver stamp mill, is really the growth of the stone mortar and pestle used by the Indians†. It does not differ in any essential particular from that described in Chapter IV., vol. i. The mills are arranged in three different ways. All the stamps are in one line, which is the usual method in all small mills, and is also the arrangement in many of the sixty-stamp mills. The stamps are placed in two lines facing each other, as in the Father de Smet Mill, in Dakota. Or the stamps are placed back to back, as in the Treadwell 240-stamp Mill, in Alaska. The Homestake with 80, and the Golden Star with 120, in Dakota; and the California Mill with 80 stamps in Virginia City, Nevada. The frames are almost universally made of wood, though when they have to be transported long distances they are sometimes made of iron, as the detached pieces weigh less, but wood, on account of its great elasticity and less likelihood of deforma-

* Trans. Inst. Min. Eng., vol. x., p. 88.

† School of Mines Quarterly, vol. viii., p. 61.

tion or fracture, is to be preferred. The battery blocks of the Father de Smet Mill are made of 2-in. planks spiked together, with broken joints clamped and bolted. This construction was adopted because it was found difficult to obtain perfectly sound timber of sufficient size to make the blocks in the ordinary way. Experience has shown that this is an excellent method, as sound planks can always be had, while large blocks perfectly sound, like those required in the ordinary stamp, are difficult to obtain, so that the composite block is more likely to insure the best results than a single stick. The stem guides are often independent, but usually in the older mills the guides are all in one piece, so that the whole battery has to be stopped when one stem is taken out. In some mills there is a rubber projection round the lower part of the guide to prevent grease falling into the mortar.

The mortars are built of different forms to suit the peculiarities of gold-bearing ores as well as their value. They are made for low, high, and overflow discharge, sometimes wide and flaring, sometimes narrow and contracted at the bottom up to the discharge opening. Their form must be determined in every case by the special conditions of the ore. Narrow mortars accelerate the discharge, and, consequently, increase the output. This is not desirable when battery amalgamation is used. The mortars should be lined on the sides and bottom with cast iron or steel plates, called "liners," to prevent the rapid wearing out of the mortar, five plates being used for the purpose, which may be easily replaced when worn. The feed openings after the liners are in should be 4 in. wide and nearly the whole length of the mortar. Generally in gold mills the discharge lip or apron should be of iron, cast as a part of the mortar, as this prevents any leakage except around the screen frame. This lip or apron should be gradually tapered to the point of discharge. It should be open and not less than 16 in. wide, which is the width of the discharge sluices, and an easy descending grade to permit of an even and shallow flow of the pulp over the copper or silver plates with which the bottoms are covered. Gold mortars are usually single discharge. They may be arranged to have copper amalgamated plates both in the back and front and sides, the

frame being bolted on to the mortar and the latter fastened under the screen frame, or they may be arranged with plates in front only, which is the method peculiarly adapted for base ores, which need to be crushed fine to separate the minerals which are afterwards to be concentrated. End copper plates are rarely used. These mortars were at one time cast very low, with the intention of adding at the mill a wooden frame to the upper part, to bring the mortar up to its usual height, but so much mercury was lost at the joint between the iron and the wood that this practice was abandoned. When the rock to be crushed is very hard, the lower edge of the screen is made high to diminish the breakage of the screens, but this decreases the output correspondingly. The mortars usually weigh from 4000 lb. to 6000 lb. when they are cast in one piece. When transportation is very difficult and the battery has to be carried upon mule back, the different parts have to be made very light, not over 300 lb. maximum weight, and the mortars are then made in several parts bolted together. The base is usually, under these circumstances, of cast iron in two sections, planed and fitted to each other, bound longitudinally by perfectly fitting bolts, and secured from side play by iron bars. The housing for about one-half of its height is made of boiler iron or steel plates, which are rivetted to the cast iron below. To reduce the weight and possible breakage, the joint is made in such a way as to be both strong for this purpose and water and mercury tight. The mortars are usually made wide when battery amalgamation is used and narrow when it is not. The object of the greater width in the first case is to give plenty of room for the plates. A well-made mortar will last four to six years. The ore is introduced through the opening in the back near the top, and is distributed under the stamps by their action, the waves being in the order of their drop. Water enters at the top against each stamp, and carries the crushed pulp through the screens as fast as it becomes fine enough, being dashed up against them by the falling stamps. Amalgamation takes place inside of the mortar on the copper plates by the pulp being dashed against them. The output of the stamp of course depends upon the fineness of the screen.

The height of the drop must be regulated by the character of

the ore, the weight and speed of the stamps. Usually a higher drop is required in gold than in silver mills, as more splash is required in battery amalgamation, and the ore must not only be crushed but amalgamated. The height usually varies from 4 in. to 9 in., 6 in. is about the mean height in California. Increase of speed always pays when it is desired to run large quantities of rock through the battery, but when the amount is limited the speed should not exceed seventy to eighty drops per minute, as some time must be given the gold to come in contact with the mercury, while eighty to eighty-five drops will generally be found to give more economical results. Each stamp is expected to crush from 1 to 3 tons of ore, according to the nature of the rock, the weight of the stamp, the height and number of drops, and the fineness of the screen used. The number of stamps in the mills varies with the quantity of work to be done, ranging from a single battery up to 240 stamps, which is the largest single mill yet built. They are almost always arranged in batteries of five, with four batteries in a set.

The battery screens are usually punched, but wire screens either of steel or brass are prepared for dry crushing, as when they are kept clean they have the greatest capacity, but wire cloth screens are more apt to clog than punched ones. Punched screens are usually made of Russian sheet iron. The slots are vertical, horizontal, or inclined. They are in parallel or broken lines, with the space between the ends of the slots in inclined or vertical lines. The slots are ½ in. long, and there are eight of them to the inch. Formerly there were only seven. The numbers of slot-punched screens runs from four to ten inclusive. The numbers of slot-cut or burred screens runs from four to twelve inclusive. The numbers of round-punched screens runs from three to six inclusive. Round punched, however, are seldom used. The numbers of either punched or cut screens correspond with each other, thus the No. 7 slot cut, as well as No. 7 round punched, have the same apertures aggregating the same dimensions and admitting an equal discharge of pulp from the battery. In wire cloth screens thirty meshes correspond to No. 7 slot, forty meshes correspond to No. 10 slot screens, and the proportion is considered as from one to four. The screen frames should

be set with an outward inclination at their top, and for gold ores of an average value the distance from the bottom of the screens set in the frames to the top of the dies in the mortar should be at least 7 in. When one edge is worn the screen is turned upside down. The screens last longest with wide mortars and high discharge. In some mills, in order to preserve a uniform height of discharge for the screen, the frames are slotted or the wooden strips on which the inside amalgamated plates are set are replaced by lower ones as the dies wear down.

The best screens are those known as "horizontal slot," with the burr left on and turned in; the slots being cut with a No. 6 chisel, and each row cut alternate, so as to break joints. This way of cutting makes them stronger and more durable, and gives a better discharge than where the slots are perpendicularly arranged over each other. From 4 in. to 5 in. in height and width of mortar discharge is sufficient; all over that causes screen to give out quicker and costs extra.

The size of the holes or meshes is very variable. In punched screens, No. 6 is the one most used, which has holes of 0.27 in. in diameter, and has 196 holes to the square inch. It is desirable,

DIMENSIONS AND WEIGHT OF NEEDLE SLOT SCREENS.

No. of Needle.	Mesh.	Width of Slot.	Russian Gauge. Thickness of Iron.	American Wire Gauge. Thickness of Iron.	Weight Per Sheet.	Weight Per Square Foot.
		in.	No.	No.	lb.	lb.
1	12	$\frac{58}{1000}$	15	$22\frac{3}{8}$	$13\frac{1}{2}$	1.24
2	14	$\frac{48}{1000}$	15	$22\frac{3}{8}$	$13\frac{1}{2}$	1.24
3	16	$\frac{43}{1000}$	15	$22\frac{3}{8}$	$13\frac{1}{2}$	1.24
4	18	$\frac{35}{1000}$	15	$22\frac{3}{8}$	$13\frac{1}{2}$	1.24
5	20	$\frac{28}{1000}$	14	$23\frac{1}{4}$	$12\frac{1}{2}$	1.15
6	25	$\frac{27}{1000}$	13	24	$11\frac{3}{4}$	1.08
7	30	$\frac{24}{1000}$	12	$24\frac{1}{2}$	$10\frac{3}{4}$	0.987
8	35	$\frac{23}{1000}$	11	25	10	0.918
9	40	$\frac{20}{1000}$	10	26	9	0.827
10	50	$\frac{18}{1000}$	9	27	8	0.735
11	55	$\frac{161}{1000}$	8	28	$7\frac{1}{4}$	0.666
12	60	$\frac{15}{1000}$	8	28	$7\frac{1}{4}$	0.666

DIMENSIONS AND WEIGHTS OF ROUND HOLES PUNCHED IN IRON OR STEEL SHEETS.

Diameter of Hole.		Thickness.		Iron.
Inches.	Millimetres.	Iron—No. Gauge.	Steel—No. Gauge.	Weight—Square Foot.
	$\frac{3}{4}$	26	28	.8
	1	24	26	1.
$\frac{6}{100}$	...	22	24	1.25
$\frac{1}{16}$	$1\frac{1}{2}$	20	22	1.5625
	2	18	20	1.875
	$2\frac{1}{2}$	16	18	2.5
	3	14	16	3.125
$\frac{1}{8}$	...	14	16	3.125
	$3\frac{1}{2}$	13	15	3.75
$\frac{5}{32}$	...	12	14	4.375
	4	12	14	4.375
	5	10	12	5.625
$\frac{7}{32}$	$5\frac{1}{2}$	9	11	6.25
	6	8	10	6.875
$\frac{1}{4}$	...	6	8	8.125
	7	6	8	8.125
	8	4	6	9.375
	9	4	6	9.375
$\frac{3}{8}$	...	3	5	10.
	11	3	5	10.
$\frac{1}{2}$	$12\frac{7}{10}$	2	4	10.625
1	$25\frac{3}{5}$	1	2	11.25
$1\frac{1}{2}$	$38\frac{1}{10}$	1	2	11.25
2	$50\frac{4}{5}$	1	3	11.25
$2\frac{1}{2}$	63	2	4	10.625
$2\frac{3}{4}$	$69\frac{3}{5}$	2	4	10.625
3	$76\frac{1}{2}$	2	4	10.625

especially with low-grade ores, to crush as coarse as the Frue vanner, or any other of the improved concentrating machines, can treat to advantage. The screens were formerly made with holes or slots of the same size throughout, but experience has shown that it is better to increase the size of the holes in the upper part rather than to have them of the same diameter throughout. Screens having twelve holes to the inch below and ten above are about the sizes used for low-grade rock.

The slots are $\frac{1}{2}$ in. long, and are set vertical, horizontal, or diagonal, as required. The plates from which they are made are 28 in. by 56 in., having 10.88 square feet in each sheet. With a moderately hard ore a stamp of from 800 lb. to 900 lb. will crush 3 to 4 tons, and those from 600 lb. to 700 lb. $2\frac{1}{2}$ to 3 tons n twenty-four hours.

The screens are keyed into the front of the mortar. They are sometimes set vertically, but usually with an inclination of from 10 deg. to 15 deg. A piece of heavy canvas is hung in front of the screen frame to serve as a splash board. The fineness of the rock is in part regulated by the position of the screen, the fineness being given by what is called "splash stamping," when there is no screen, but the pulp is delivered over the front of the mortar.

Gold mills are always wet crushing. The amount of water used will vary somewhat according to the quantity available. It will generally not be less than from 1000 to 2000 gallons per ton. Some mills use more than this, but the amount will generally be between 1500 and 1800 gallons per ton. In the winter the water must be heated by direct steam. It should, however, never be made more than tepid, as a higher temperature makes the mercury too lively. When water is scarce it can be used over again.* There is a great variety of opinion as to the best arrangement of the drop of the stamp in order to secure a uniform splash and flow or wave of the water and pulp in the mortars. This has already been discussed.† A very good order for gold mills is Nos. 1, 4, 2, 5, 3, the effect of which is that No. 1 throws the water through the whole length of the battery towards No. 5, No. 4 catches this wave in its descent upon it and throws it back, while No. 2 catches it, throwing it back again until No. 5 catches it, and the fall of No. 3 equalises the wave. In theory this is correct, and produces a regular wave in the mortar from the ends to the central point, but there are other orders of discharge which answer perfectly well. The dropping of the stamps consecutively in their numerical order should be avoided, as this causes an accumulation of pulp at that point; hence the order 1, 5, 2, 4, 3, is not a good one, as 3 and 4 drop consecutively. The battery feed water should always flow in at the back of the mortar with the ore. Formerly it was arranged to flow in at the front, which prevents a perfect wave and hinders to some extent a perfect discharge.

It is most remarkable how the screens wear. They last from two to three weeks or longer, and are frequently capable

* Vol. i., p. 442 † *Ibid.*, p. 180.

of bearing the discharge of over 4000 tons of comminuted rock before they have to be replaced. Fine stamping with slow speed is more advantageous when blankets are used than rapid, as the latter makes more float than the former. When plates are used the stamps are run at ninety to ninety-six drops with low discharge, so that but little slime is formed. The water should be fed in just sufficient quantity to carry the pulp out of the battery and keep the plates clean. When the screens are very fine, as high as sixty, part of the gold and mercury is either flattened or floured, and then no plates will catch them. The ore should never be crushed so fine that the pyrites cannot readily be caught on a concentrator. There would seem to be an advantage in doing the stamping in hot water, for fine particles of gold will remain floating on cold water at the ordinary temperature for over twenty-four hours, but when the water is boiling they sink quickly. It would probably not be possible to do this on any very large scale, as it has been found by experience that very hot water makes the mercury too lively.

The cams are lubricated with a mixture made of half a gallon of whale oil, 2 lb. of tallow, 2 lb. of beeswax, 3 lb. of resin, and ½ lb. of lampblack. This paste is too hard for use cold, and must be heated when it is to be used. It is kept in cans and put on hot with a mop stick. The necessity of lubrication is always known from the too rapid revolution of the tappets. The stamps are lubricated by a similar mixture, made softer by the addition of less lampblack and resin. This lubricating mixture will not affect the amalgam, if a little of it gets into the mortar, but it should not be allowed to get there if it is possible to avoid it. The cams are often lubricated with soap or a mixture made principally of treacle, as this is soluble and does not affect the mercury. Whatever mixture is used, the greatest care must be exercised that no grease of any kind gets into the mortars or on to the aprons or sluices. Absolute cleanliness about a gold mill is an indispensable requisite to success. Axle or other grease, when it is used for any purpose, such as lubricating the shafts, should be used but little at a time, and should be put on with a swab for fear of using an excess.

The setting of cams upon a cam shaft to replace one that had

been broken is a very difficult matter, especially in small mills

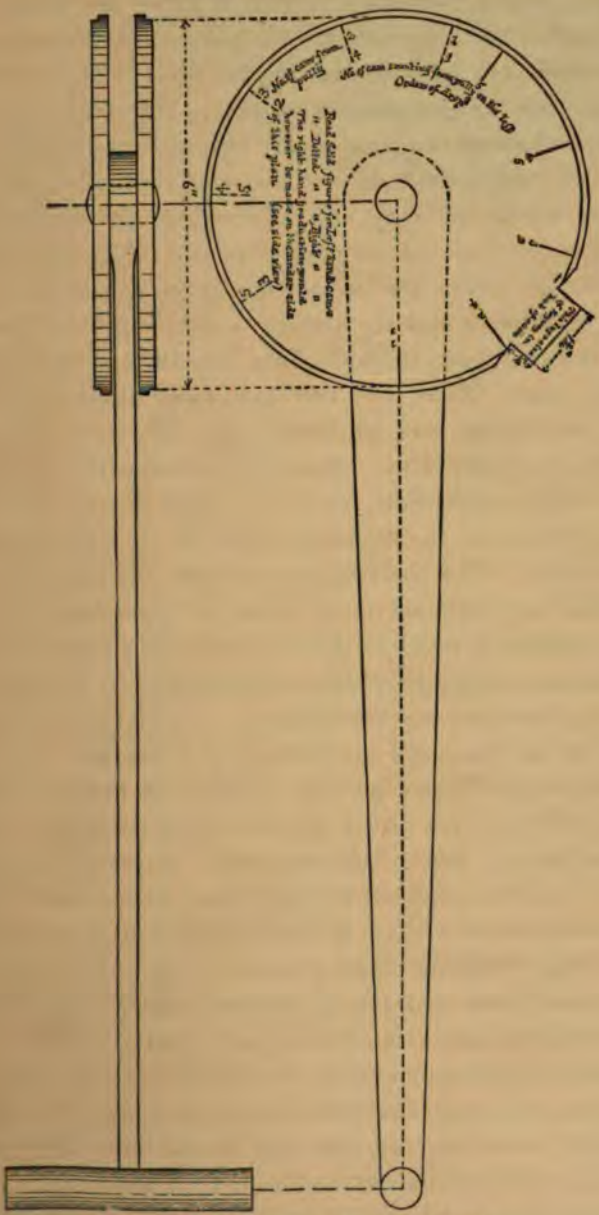


FIG. 49. CAM KEY SEAT PROTRACTOR.

where there is no machine shop and but few tools. A protractor,

Fig. 49, which is used in many of the mills in the West,* the drawing of which is made to a scale of about one-third of the actual size, has been invented for the purpose of determining the exact position of the key seat in the cams. This protractor enables any one to determine the position of any of the key seats of the cams, and to set the cams of any two batteries in such a way that no two stamps shall drop at the same time, which results in giving the maximum work and efficiency to the mill with the expenditure of the least amount of power. The protractor is made with the top of the circles exactly the size of the hole in the hub of the cam and the arm the exact length of it, so that when it is placed upon the hub of the cam the end of it will rest upon the nose. There are two graduated circles above and below, used for right and left-hand cams. The circles are graduated both above and below. For left-hand cams the circle above is used, and for right-hand cams the circle below. The right-hand graduation on the drawing should be on the under side of the protractor. The solid figures are for left-hand cams, and the dotted ones supposed to be below for right-hand ones. To use the protractor, place it on the hole in the hub of the cam with the proper side up and move the circle until the number of the cam to be cut corresponds with the zero division of the arm. The lug on the side will give the proper position of the cam slot. The outside figures give the number of the cam and the inside the order of the drop. Fig. 49 shows the proper position for the keyway of No. 1, left-hand cam. When the position of the cam has been marked, the protractor is removed, and the keyway may be cut with a slotting machine if there is one at hand, or with a drifting chisel if there is not. This last method is a very simple one and can be used anywhere. Fig. 50 shows a solid cast-iron hub with a handle and a slot cut into it. The lower part of this hub exactly fits the hole of the cam. The upper part sets upon the hub, fitting it tight. The slot for the keyway is cut out on one side of the hub. The drifting chisel, Fig. 51, is then driven with a heavy hammer, with a $\frac{1}{16}$ in. drift, using tapering keys in the keyway hole in the drifting

* For this drawing I am indebted to Mr. G. H. Robinson, of the Drumlummon Mill, Marysville, Montana.

hub behind the chisel. After the first cut a $\frac{1}{16}$ in. shim is put in the keyway back of the chisel, and the chisel driven through

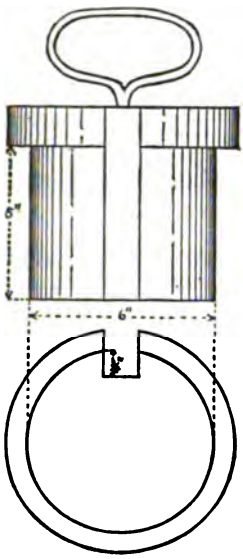


FIG. 50. DRIFTING HUB.

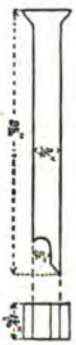


FIG. 51.
DRIFTING CHISEL.

for a fresh cut. This is repeated until the keyway is of sufficient depth. This insures as much accuracy as a machine, and, if properly done, saves a great deal of labour, and allows of any machinist cutting the cam keys in a very short time with very little expense. When the cams are in two pieces screwed together, the repairs may be made in this way in a very short time, as a single cam can be removed at a time; when they are solid the whole battery must be stopped to put one on.

Breakdowns are apt to happen at any time, especially with mills of heavier than 300 lb. stamps, and the millman should see to it that supplies of all kinds likely to be required are on hand ready to use, and that tools are kept in their places when not in use; shoes with their wooden slips tied on ready for use, slips and strings in plenty, strips of gunny-bagging, tappet and cam keys, liners, duplicate screens and parts likely to break down or wear out. If a shoe drops off it should not take over ten minutes to hang up the five stamps of this battery, take out the screen, replace the shoe, and go on again. It has often been done in half that time. The same may be said in regard to a broken

shoe or die. A slipped tappet may be reset to its mark without stopping the other four stamps, by simply hoisting up by Spanish windlass or by Weston pulley, and adjusting it; but if a stem or cam breaks, or a cam gets loose, after half the week is passed, it is better to hang the defective stamp up out of the way of the other four until Sunday, unless it can evidently be replaced within an hour, as all the stamps working on that shaft must stop while this is being done. This refers to the ordinary solid cams, but sectional cams made in two pieces, which are fastened together by a stirrup and key, do not require the removal of the cam shaft. They can be taken off and adjusted in half an hour or less. If a screen bursts or gives out, that battery can be hung up and the screen replaced with a duplicate; the old screen may then be repaired and put away until a like emergency requires another. Dies can be used until they are worn down to the flange, and shoes may be worn to within 1½ in. of the head before renewing, unless broken before that. Steel shoes and dies rarely break, and wear down slowly; this wear lightens the stamp, but not enough to justify replacing it until it is in danger of breaking. Some of the modern-forged steel shoes, as at the Drumlummon Mill, wear until the head breaks off from the stem of the shoe as a ring. The chrome-steel shoes at the Stickels Mill have lasted eight months, and were good for two months more. An iron shoe generally lasts from one to two months, steel shoes usually last from two to three times as long as iron ones and cost about twice as much. Iron shoes are most generally used, as they can be made in the local foundries, while steel shoes must be brought from a distance, which, when transportation is difficult, increases the cost. When the tappets are set the stems should be marked with chalk, so that any slip may be at once detected even before the unequal noise of the stamps draws attention to it. With the use of the modern steel shoes and dies, tappets need only be reset once a week. The best way to reset tappets is to have five blocks cut the exact length of the required drop, and to put these under the stamps while setting the tappets on the stems to suit the cams. The keys must then be driven in hard, so as to secure a firm grip on the stem. The drop of the stamp must, of course, be in accordance with the hardness of the ore being crushed.

Ordinary quartz of a saccharine or marbly structure requires to be set with a 7-in. block; softer, 6 in.; harder, 8 in. or 9 in. More than this has been found to do but little more work, while it wears the shoes and the dies out faster, and consumes extra motive power, much more than the small increase in production justifies. The number of drops per minute must be so regulated that the stamp has finished its fall before the cam strikes the tappet for the upward stroke. Constant vigilance on the part of the watchman is imperatively demanded to guard against accidents of all kinds, as tappets will slip, cams work loose, shoes or heads drop off, stems break, or nuts get loose and drop off into the mortar, which will cause constant and vexatious delays in the work. It must be his special care to look out for loose bolts and nuts; especially should those on the guides be kept from getting loose and falling off. A good millman is always known by his record of hours run in the week.

The following Tables give the dimensions of some of the principal mills of California and elsewhere.

Number and Quality of Mill.	Water or Steam Power.	No. of Stamps.	Weight of each Stamp.	Drop of Stamp in Inches.	Drop of Stamps per Minute.	Number of Tons Crushed per Stamp in 24 Hours.	Numbers of the Screens.	Method of Working.	Concentrators.
CALIFORNIA.			lb.						
COLUMNE Co.									
Isby . .	Water	15	750	6	85	1	60 brass wire	B	D
tz . .	,,	10	650	6	85	1	9 slot	B	C
hanan . .	Steam	20	850	7–7½	90	2	40 brass wire	B	8 F
CALAVERAS Co.									
kels . .	,,	20	950	7	86	3¾	60 brass wire	B	C
ca . .	,,	60	900	5	10	2½	9 angle slot	B	8 F
p Lead .	,,	8	750	9	85–87	Carloads, 5	5 mesh		
ep Ranch .	,,	30	800	8	85	2¼	9 angle slot		
e Gold . .	Water	10	750	6	100	2 1/10	40 mesh		
Plata . .	,,	15	750	6–8	96	2 6/10	5 slot		
. . . .	,,	40	850	7	95	3–4	30 brass wire	B	8 F
AMADOR COUNTY.									
Amouth Consoli-ated . .	,,	80	750	7	90–100	Av. 2	8 round	B	32 F
Amouth Consoli-ated . .	,,	40	1000	7	90–100		8 ,,	B	16 F
rstone . .	Both	40	750	6	93	2½	8 straight slot	B	28 E
th Spring Hill	Water	30	750	6–7	90	2½	7 angle slot	B	10 F

Number and Locality of Mill.	Water or Steam Power.	No. of Stamps.	Weight of each Stamp.	Drop of Stamp in Inches.	Drop of Stamps per Minute.	Number of Tons Crushed per Stamp in 24 Hours.	Numbers of the Screens.	Method of Working.	Concentrators.
AMADOR CO.—*con.*			lb.						
Stewart . .	Water	40	850	7	85	4½	6 angle slot	B	C
Moore . .	,,	10	850	7	90	3	6 ,, ,,	B	C
Downs . .	,,	20	600	8	85	1¼	9 and 10 slot	B	C
Mammoth Tunnel	,,	10	750	8	90	2½	6 slot	B	
Zeile . . .	Both	40	750	7½	87–88	3 37/100	4 straight slot	B	16 F
Bunker Hill .	,,	40	850	6½	90–92	2½	8 angle slot	B	16 F
EL DORADO CO.									
Springfield .	,,	15	600	9	90	1¾	...	B	6 F
PLACER COUNTY.									
Dardanelles .	Water	5	850	7½	95–97	4½	3/16 round		
Live Oak . .	,,	5	1100	7½	95	12	1/2 ,,		
Paragon . .	,,	10	900	9	100	9	1/2 ,,		
Morning Star .	,,	10	900	9	68	5	3/16 ,,		
NEVADA COUNTY.									
North Star .	,,	30	850	7	90	2	30 mesh	B	12 A
Idaho . .	,,	35	850	9	72	2 85/100	6 slot	B	C
Empire . .	,,	40	850	7	90	2	30 mesh	B	16 A
Providence .	,,	40	750	6–8	93	2	5 round punch	B	16 F
Con. Wyoming .	,,	16	800	7	92	2	...	B	6 F
Nevada City .	,,	10	850	6	85	2	6 round punch	B	2 F
Delhi . .	,,	8	1000	6	94	3½	5 ,, ,,	B	2 A 2 F
Eagle Bird .	,,	20	750	6–7	95	1½	40 mesh		
SIERRA COUNTY.									
Sierra Buttes .	,,	60	850	8	86	2⅓	7 slot	B	24 F
Young America .	,,	40	750	7	80	2	7 ,,		
Alaska . .	Steam	40	850	7	86	2¼	6 ,,		
MONTANA.									
Drumlummon .	,,	60	750	7–9	96	2½	30 steel wire	B	24 F
Jay Gould . .	,,	10	800	5–6	90	3	4 slot	B	4 F
NEVADA.									
California Mill .	Water	80	1000	7½	95	4	2 & 3 diag. slot	...	D
ALASKA.									
Treadwell . .	,,	240	900	8	90	2½	8 slot	B	96 F
DAKOTA.									
Homestake .	Steam	80	850	8¾	85	4½	6 slot	B	D
Golden Star .	,,	120	850	9	85	4	6 ,,	B	D
Highland . .	,,	120	850	9	87	4	9 ,,	B	D
Deadwood . .	,,	80	850	9	85	4	6 ,,	B	D
Caledonia . .	,,	80	850	12	74	3	6 ,,	B	D
Golden Terra .	,,	80	850	9	85	4	6 ,,	B	D
Father de Smet	,,	100	850	8	85	4	30 wire	B	D

A, Triumph concentrators; B, Battery amalgamation; C, Cornish buddles; D, Sluices; E, Hendy's concentrators; F, Frue vanners.

The following Table gives the weight of the parts of some of these stamps:

Name of Mill.	Weight of Stamp.	Weight of Stem.	Weight of Tappet.	Weight of Boss.	Weight of Shoe.	Shoe's Lost Days.
CALIFORNIA.						
CALAVERAS COUNTY.						
Stickels . .	950	370	150	250	180	300*
Ilex . . .	850	400	120	200	110	
AMADOR COUNTY.						
Keystone . .	825	300	125	275	125	
South Spring Hill	750	240	100	250	160*	
Stewart . .	850	350	110	225	150	
Bunker Hill . .	850	350	110	225	150	
Moore . . .	850	350	110	225	150	
NEVADA COUNTY.						
Idaho . . .	850	350	160	200	140	
North Star . .	850	350	160	200	140	
Empire . .	850	350	160	200	155	150
SIERRA COUNTY.						
Sierra Buttes .	850	365	195	180	160	
ALASKA.						
Treadwell . .	900	350	140	250	160	150*
DAKOTA.						
Homestake . .	850	350	120	240	140	42
Golden Terra .	850	350	125	250	125	60
Father de Smet .	850	350	130	200	135	
Caledonia . .	850	320	125	275	140	

The ore-feeders should be set in line with the batteries, so that the action of the tappet upon the rod leading to the ore-feeder lever shall be direct, and transmit a direct action to the lever bar. Great care and constant attention should be given to the adjustment of the stroke of the tappet, in order to prevent any possible irregularity in the feed. This is a matter deserving much study, for upon the regularity of the quantity of ore delivered to the stamps the perfect or imperfect reduction and amalgamation of the ores and the amount reduced and amalgamated depends. The quantity of ore to be fed to the stamps is regulated by its character and value. The main object of quartz milling is to stamp the largest number of tons within the least period of time, provided that a maximum percentage of the value of the ore shall result. This requires attention and study, but

* Chrome steel.

if a general rule can be stated, it is, that the feed should be low, as thereby the effect of the stamp is more direct and its crushing capacity greater.

There should always be a clean-up room, as convenient to and as near the central point of the concentrators, pans and bateas, as is practicable, that no loss of amalgam or quicksilver shall occur in the process of cleaning it. It should be easy of access, well fitted up with sinks, trays, shelves, and arrangements for the supply of an abundance of clear water. It should contain two tanks of wood or iron, 4 ft. by 3 ft. and 3 ft. deep, situated near the light, and kept constantly filled with water to prevent losses in panning. The floor should be mercury tight, and should incline slightly to a catch basin. Cement is on the whole the best material for the floor. The character, variety, and quantity of testing and cleaning up utensils is a matter which varies in almost every mill, and depends upon the fancy and taste of the individual fitting it up. The mercury in the mill should be moved automatically; the method of doings this by pumps has been previously fully described.* Of late years it is very frequently done by the pressure of water.

The capacity of a gold mill varies from fifty to several thousand tons a month, according to the size of the mill and the quantity and hardness of the ore. Some of the largest of these in the United States are the six Homestake mills in Dakota, which drop five hundred and eighty stamps. There are three mills of eighty, two of one hundred and twenty, and one of one hundred stamps. The largest single mill is the Treadwell, on Douglas Island, Alaska, which has 240 stamps. It has been found by long experience comparatively less expensive to run a large mill than a small one. The expense of running one hundred stamps is but a little more than that of running fifty, and the whole cost of the ore treatment relatively much less. It has also been found that the quantity rather than the quality of the ore must be depended upon for successful working. Stamps were formerly run almost exclusively by steam. Water-power, if it was depended upon at all, was regarded as being very uncertain, and the conditions under which it could advantageously be used could only be

* Vol. i., p. 397, *et seq.*

occasionally found. Steam was always made an adjunct to a mill run by water-power, to provide for the accident of the giving out of the water, which was always likely to occur. Wood is the fuel generally used. When the mills are at the bottom of a ravine, as the Homestake Mills, the wood is shot down in a shute set at an angle of 60 deg. from a height of 200 ft. or more above the mill. The bottom of this sluice is lined with sheet iron, and sometimes a little water is allowed to flow in the bottom of it to prevent its taking fire from friction. About 20 ft. from the lower end the trough flattens. The extreme end is movable, and is turned up at such an angle as to throw the sticks to the part of the pile where the wood is to be stored. As the inclination and the side angle can be changed at will, the pile can be kept even. The transportation of the wood in V water troughs is also quite common. Within a few years the modern methods of the use of water at high pressures from mining ditches, rendered more or less idle by the débris litigation, has rendered the use of water for power practicable under such circumstances as in some places to supersede the use of steam altogether. Each stamp will generally require from two to three horse-power, when steam is used, but this will depend very largely upon the number of stamps, a small mill requiring proportionally more horse-power to the stamp than a large one. The pressure of steam ordinarily used is about 80 lb. Occasionally it is 90 lb. or even as high as 100 lb. Very few boilers are run with a pressure of less than 70 lb. Under the present conditions, in California especially, water-power on the ditch lines is more economical than steam. Some mills are fitted with both steam engines and water wheels, the former being brought into use when the latter are unavailable in a case of drought or freezing. This last difficulty seems unnecessary in view of the practice in the Dakota mills, which has already been described in the chapter on hydraulic mining. The water motors generally used are Lefel or Knight's turbines, overshot wheels, and the Pelton and Knight's patterns of impact or hurdy-gurdy wheels.

The success of the Idaho Mine in Grass Valley, in substituting in the year 1883 water from ditches in place of steam, in which

they realised in useful work 80 per cent. of the power exerted, at a cost of $46,500, has led to the organisation of a company to furnish water-power to the mines at Union Hill, Ophir Hill, and others, and to its very general use for power in California. The water for the Idaho Mill is furnished by the South Yuba Canal Company from the reservoir at the foot of Banner Hill. Pipes 25½ in. in diameter carry it to Union Hill, a distance of 2½ miles, with a fall of 400 ft. At Ophir Hill it connects with the Magenta Mine, and at Wolf Creek with a series of mines, some of which have a pressure of 500 ft. The water can be used so much cheaper for power than steam that it is expected it will not only be used for prospecting mines but for other purposes. In a few cases, when there is a great difference in the elevation of the mills, the water used at one mill can be used over again at another. This is the case at the Empire Mill in Grass Valley, where the water leaving that mill is carried to and runs the North Star Mill.

Each stamp is estimated to crush two to four tons in twenty-four hours, according to the hardness of the rock, or for a 20-stamp mill 1200 to 2500 tons per month. The cost of water-power per ton of ore stamped will, therefore, be about 20 cents. The rate for the miners' inch is 20 cents, so that it may be estimated that the capacity of the inch is about a ton.

The Table* below gives the amount of water-power used in some of the mills of California, and its cost.

——	Number of Stamps.	Cost of Water Power per Month.	Pressure in Feet.	Use
		$		
Lincoln . . .	40	403	270	For milling.
Keystone . . .	40	1200	260	Hoisting and milling.
Bunker Hill . .	20	903	260	Milling.
Gover . . .	30	550	130	
Oneida . . .	60	1500	225	Hoisting and milling.
Kennedy . . .	20	450	170	,, ,,
Amador Consolidated	60	600	470	,,
Coney . . .	40	400	...	Milling.
Mahoney . . .	40	400	...	,,

All of these mines, with the exception of the Keystone, do not yield over $10 to the ton.

* Prod. Gold and Silver in the U.S., 1880, p. 339.

There is much difference of opinion with regard to battery amalgamation, the objections to it being that it is a constant temptation to steal; but if 75 per cent. of the gold can be caught in the battery, as is done in some of the mills by the addition of 15 oz. to 20 oz. of mercury in twenty-four hours, the question is asked by those who are in favour of it, why not catch it there and leave the other 25 per cent. to be caught elsewhere? There seems to be no doubt about battery amalgamation when the gold is in large pieces, for these settle in the mortar, and by constant attrition with the pulp become bright, and then are caught by the mercury. In such a case, as most of the gold caught on the outside plates is fine and has passed the screens before it has been attacked by the mercury, it would seem to be best to run with a high drop, low speed, and a deep mortar. It is very doubtful whether it is ever best to amalgamate in the battery ores with a very large percentage of sulphurets.

The width of the mortar at the bottom of the screen is usually from 10 in. to 14 in. It varies with the hardness of the ore. With very friable ores it may be as narrow as 10 in., but with hard ones it must be wider for fear of injuring the screens. This may be obviated by raising the height of the discharge, but this can only be done at the expense of the output. The narrower the mortar is, the faster the screens will discharge. Narrow mortars are not always desirable when battery amalgamation is used, as it diminishes the time that the gold remains in contact with the quicksilver. Double discharge mortars are very rare in gold mills. When the ore contains such a large percentage of sulphurets that mercury cannot be used in the battery they may be advantageously used, as the ore remaining too long a time in the mortar is likely to be crushed too fine, and the free gold made into "float," or rendered difficult to amalgamate; but when battery amalgamation is practised they are necessary.

When amalgamated plates are used in the battery there is but one discharge, as a single screen will discharge the ore as fast as it can be amalgamated. In order to amalgamate as much gold as possible, the level of the top of the die is placed from 2 in. to 6 in. below the discharge. To keep the discharge always at the same height as the die wears out, the screen has a movable slide which can be lowered.

The amount of quicksilver to be charged can be determined only by the gold value of the ore passing through the batteries, and is regulated by the "feel" of the amalgam accumulating on the outside plates, which should be tried from the battery apron to the lower edge of the last line of sluice plates. It is necessary to add quicksilver enough to amalgamate the gold and still keep it in the pasty condition which causes it to adhere to the copper plates. Fine gold requires more than coarse, and ores which contain a high percentage of sulphurets more than those which do not. Such ores have a tendency besides to scour the plates as well as to divide the quicksilver. If the amalgam feels hard and crumbles at the touch, then the quantity of quicksilver is not sufficient, and a larger amount should be charged; if, however, the outside amalgam is found thin and slippery, and some loose or passing away in small globules, then a less quantity must be charged. Careful observation alone can regulate this important matter. A thorough scrutiny of the character of the ores as they are fed to the batteries should often be made, for frequent changes in their value will occur. The ores should be often "horned" or prospected, as a guide to their value. A good amalgamator can determine the value of an ore by a horn prospect as well as a professional assayer. In ores of medium or low-grade value a charge of quicksilver, varying from ¼ lb. to ½ lb. in each battery in twenty-four hours, should be made every hour, with rich every half-hour. A fair proportion for charging should be between 2 oz. and 3 oz. of quicksilver to 1 oz., troy, of gold per ton, when the gold is from 750 to 800 fineness, or valued at from $15.50 to $16.50 per ounce; that is, clean amalgam prepared for the retort can usually be valued at, say, $6 per troy ounce when the gold is of the above fineness, and generally loses two-thirds of its weight in retorting, *i.e.*, 60 oz. of clean amalgam should produce 20 oz. of retorted amalgam, 750 to 800 fine, worth from $15.50 to $16.50 per ounce net in coin at the United States Mint. In any case, enough mercury must be put in to insure the gold being put into that pasty condition in which it adheres to the plates. The quicksilver is usually fed into the batteries every half-hour, being added little by little by hand. An automatic feeder has been introduced, so arranged that the cam shaft moves a ratchet wheel, which is made to revolve in

one, two, three, or four minutes, which dips a small cup into a quicksilver reservoir and empties it into a pipe leading into the batteries, making the feed both regular and constant. In some mills, as at the Homestake, all the mercury is fed in the mortar. At the Caledonia part is added in the mortars and part on the apron plates. The method best adapted to the particular ore must in every case be settled by an actual trial.

The amount of gold caught in the batteries varies from one-third to two-thirds of that produced as amalgam. It is generally about one-half. The following Table gives the amounts of gold caught in some of the California mines:

	Caught in the Battery.	Caught on the Sluices.
Kelz	85	15
Soulsby	90	10
Buchanan	66	34
Sheep-ranche	80	20
Fine Gold	50	50
Abbey	80	20
North Star	80	20
Empire	80	20
Providence	20	80
Wyoming	40	60
Delhi	33-40	66
Eagle Bird	50	50
Sierra Buttes	73	27
Young America	95	95
Alaska	65	35

The more silver the gold contains, and the finer it is, the more mercury it requires. The bullion from the battery is generally worth from $12 to $18 per ounce, showing that it contains but a small amount of silver. Only a little lead and copper is found in such bullion. Gold is not so perfectly alloyed with the mercury as silver. It seems to be merely coated on the outside, and this coating causes the grains to adhere, but when pressed between the fingers the grains can be felt. For this reason it is necessary to keep the amalgam of a given consistency. If there is too much mercury, it will flow too easily, and will not be caught on the plates, and if too little the gold is not sufficiently amalgamated and is lost in the tails. It is not always easy to ascertain the exact quantity of water used in the batteries, but it is probably between 500 and 700 gallons per ton of ore crushed. According to the data I obtained in San Francisco,* each stamp

* J. M. Scott, Union Iron Works, San Francisco.

requires 10 lb. per minute, which, at the rate of 3 tons crushed per stamp, makes 576 gallons per ton. Many mills use more. Where water is scarce it may be used over again from the slime tanks or ponds to the battery.*

The current of the water feed must not be strong enough to wash and scour the plates when issuing from the mill, nor so slow as to allow it to become thick and turbid, and to lodge and settle on the plates. It should issue smoothly, and a little experience will soon teach how to regulate it for the different kinds of material put through, which varies so much as sometimes to make a hundred per cent. difference in the flow of water required.

The loss in quicksilver is generally greater in cold than warm weather, especially in gold mills. It appears to be a well-established fact that in all phases of its amalgamation it is greatly facilitated by high temperature and is interfered with by cold. The loss, however, is very small. It was found at the Homestake Mill that it amounted to 5.27 lb. a year, or 0.0044 lb. per ton of rock crushed. At the Caledonia Mill it was 7 lb. a year, or 0.0011 lb. per ton of rock crushed.

The pulp after it passes the battery is treated in different ways. As it contains considerable gold amalgam, when amalgamation is practised in the battery, it is made to flow over an amalgamated copper plate, set 6 in. to 8 in. below the lip of the mortar on which most of the quicksilver alloy is caught. This plate is called the apron. It is as wide as the battery screen and usually from 3 ft. to 5 ft. long. It varies from 30 in. to 15 ft. in length, the length depending on whether amalgamation is done in the battery or not. It is generally covered with silver-plated copper plates, $\frac{1}{8}$ in. thick, fastened with plated copper screws. Usually there is a single drop of 6 in. to 8 in. from the mortar lip to the aprons. At the Bunker Hill Mill the lip of the mortar discharges with a drop of 4 in. to 5 in. on to a copper plate, inclined away from the mortar at an angle of about 20 deg. This discharges on to another about the same angle inclined towards the mortar. From the end of this it discharges with a fall of about 6 in. on to the battery apron. These plates are the width of the mortar and 8 in. long. The result of this constant

* Vol. i., p. 442.

impact is that little or nothing is found beyond these plates, 90 per cent. of the gold caught being taken from them. The grade of the sluices from the battery to the settling tanks is usually from ¾ in. to 2½ in. per foot; 1 in. to the foot is generally sufficient except with heavy ores, and more than 2½ in. is probably never necessary. Where high grades are employed, as they often are for short sluices, it is not because they are necessary, but because the disposition of the apparatus makes it convenient. They are followed by sluices which vary in width from 14 in. up to 22in., with inclinations of ¾ in. to 2 in. to the foot. The total length of some of the sluices, as at the Young America mines, is 800 ft. At the Sierra Buttes it is 240 ft. The sluices are generally narrower than the aprons, which is a bad plan, as the force of the water is thus increased, and the fine particles of of gold are more likely to be carried off. The sluice plates are generally four in number, and are screwed down, the upper overlapping the lower plate. Sometimes all the plates are brazed into one long plate. The following Table gives the dimensions of the aprons of some of the prominent mills:

Name of Mill.	Size of Apron Plates in Feet or Inches.	Length of Sluices in Feet.	Width of Sluices in Feet or Inches.	Grade of Sluices in Inches to the Foot.
Stickels . .	14 ft. by 5 ft.	12 ft.	24 in.	3 in. in 10 ft.
Ilex . . .	5 ft. 8 in. by 5 ft. 6 in.	15 ft. 6 in.	22 ,,	3 in. ,, 10 ft.
Keystone Consolidated . . .	48 in. by 14 in.	80 ft. for 40 stamps	14 ,,	3½ in. ,, 10 ft.
South Spring Hill .	28 in. ,, 48 in.	128 ft. for 20 stamps	15 ,,	
Zeile . . .	30 in. ,, 58 in.	12 ft. for each battery	14 ,,	1¼ in.
Spring Valley .	2 ft. by 4 ft. 2 in.	12 ft.	12 ,,	3 in. in 10 ft.
Sheep-ranche .	4 ft. by 10 ft.	100 ft. for 30 stamps	15 ,,	2 in.
Bunker Hill . .	3 ft. by 4 ft.	12 ft.	14 ,,	3 in. in 10 ft.
Fine Gold . .	4 ft. by 4 ft. 6 in.	100 ft. to each battery	16 ,,	1¾ in.
Providence . .	4 ft. by 4 ft.	14 ft. to each battery	16 ,,	1¾ in.
Nevada . . .	4 ft. ,, 5 ft.	12 ft. to each battery	14 ,,	
Sierra Buttes. .	4 ft. ,, 4 ft.	240 ft. for 60 stamps	15 ft.	1½ in.
Alaska . . .	30 in. ,, 52 in.	14 ft. to each battery	24 in.	
Young America .	48 in.	800 ft. for 40 stamps	15 ,,	
Treadwell . .	14 ft. by 6 ft.	12 ft.	24 ..	3 in. in 10 ft.

These plates are cleaned up every morning, after which mercury is scattered over them, and with a rag or hand-broom spread out uniformly over them. When they require scraping to remove the amalgam, it is done with old files bent at right angles and sharpened like a chisel. The plates must not be scraped to the copper. When they get hard they are carefully scraped down to the copper and sent to be redressed and replated. In most places it is cheaper to have this work done than to do it. When the plates can no longer be used they must be sweated to remove the mercury and amalgam before being sold.

The pulp is then discharged as tails into sluices, which are 8 in to 16 in. wide and 2 in. to 3 in. deep. In these sluices are the amalgamated silver and copper plates, alternating with riffles or boxes for catching the quicksilver and amalgam. Passing these in some mills they fall into sluices, in the bottoms of which blankets are laid, as in the Stickels and Caledonia mills. On these blankets the sulphurets, which are usually rich in gold, are caught, and from here to the concentrators the sluices are generally made of wood, but of late years they are being built entirely of iron. The frames of the sluices should never be built in contact with the mortar or battery frames, but independent of them and resting upon independent bearings, so as to prevent jar and leakage. After leaving the blanket sluices, the tailings sometimes run into a concentrator, in which the remaining contents which are of value are recovered. The Hendy concentrator is the one which was formerly the most employed in California. It is usually placed after the plate sluices and before the blankets. Buddles are also used. Within a few years Frue vanners, more than 700 of which are in use, 200 in California alone, have been introduced. The Embrey and the Triumph concentrators have also been used. After passing the concentrators, the tails are generally allowed to run to waste. Sometimes when the amount of gold-bearing pyrites is large, they are concentrated a second time. Generally no reduction in the size of the ore containing the gold is made after it leaves the stamps, but when the pyrites are very rich in gold they are sometimes ground and amalgamated in the arrastra or treated by Plattner's method to extract the gold they contain.

When the blanket system is used and the pulp is treated on

any of the improved concentrating machines, only two lengths of sluices, each about 4 ft. long, and two sets to each battery of five stamps, should be used. These trays or sluices or aprons, as they are called, should be 16 in. wide and arranged so that the inclination may be altered from 3 deg. to 5 deg. to suit the pulp being treated. Occasionally, as in the Providence Mine, they are as high as from 8 deg. to 9 deg. The fall from the battery box should be about 4 in. for the head sluice, or the same as from one sluice to another. A hood is placed over the end, so that the pulp will strike the blanket nearly at right angles. The blanket should be made of coarse wool with a long nap. It is not so much the length of the blankets that separates the gold from its gangue, as the fall or jump on to them. When water is scarce a concentration of the tails can be made in the "Tye," and this sent to the Frue vanner. When amalgamated plates are used, they should be 3 ft. long, 2 ft. wide, and set on a frame and at an inclination of 3 deg. They are either brazed together so as to make one length, or the upper plate is made to lap over the lower one To this frame a slow lateral movement backward and forward of one foot with a slight shake may be given. Clear water should be sprinkled on it from a perforated pipe, which should fall on the upper part where the pulp strikes. The concentrates are dried, sifted, and ground in the arrastra or Chilian mill. The grade given to blanket sluices is usually from ¾ in. to 1 in. per foot, depending on the specific gravity and the fineness of the ore and the proportion of sulphurets contained in it. Sometimes the grade is increased to as much as 3 in. to the foot. The apron used in gold mills has always more grade than the sluices, and in many mills there is an arrangement by which the end of the apron can be raised or lowered as different peculiarities of the ore may demand.

The riffles which are used are of various kinds, according to the nature of the work to be done and the special traditions of that particular part of the country. Round riffles are made by boring 1½-in. holes in 1¼-in. plank, four holes being bored at the corners of every 6 square inches and one in the middle, and so on through the whole length of the plank. These are placed on the bottom of the sluice, which is generally 15 in. to 16 in. wide. They very

readily catch all the heavy pieces, and are soon filled with small pebbles and the small pieces of gold. When these planks so bored out are placed in a sluice together, a space of about 1 in. is left between every two boards, which serves the purpose of a transverse riffle.

Hungarian riffles consist of slots, arranged very much like a window blind, with the inclination towards the bottom of the riffle. These are usually made in a frame, which is covered with sheet iron to protect it, sixteen of the riffles being placed in each frame. Sometimes these Hungarian riffles are made of cast iron. They answer extremely well, and are very much used in drift mining and also in river bed mining.

The ordinary transverse riffle is used in two ways. It is either placed permanently on the bottom of the sluice with mercury behind it, or else it is made so that, as in the treatment of tails, riffles may be piled upon riffles to the depth sometimes of 12 in. to 16 in.

Longitudinal riffles, strapped or not strapped with iron, are frequently used, being made in sets with a transverse riffle at the end of every set. There may be three or four of these in the width of the sluice, depending upon circumstances. Riffles in the shape of a V, like those used at Iowa Hill, are also sometimes used at the end of the battery sluices.

The process for the extraction of the gold from vein matter may very generally be divided into distinct methods, each of which varies very greatly in its detail in different mills, but they are essentially alike. They depend upon the separation of gold by mercury, and only differ in their way of doing it. The first of these is stamping and amalgamation without concentration, as in the mills of Dakota, the best examples of which are the Homestake, Caledonia, Father de Smet, and Deadwood Terra mills in Lawrence County, Dakota. The second of these processes is known as the battery amalgamation and concentration process, as amalgamated plates are placed in the batteries, and concentrators are used, as at the Keystone Consolidated Mine in Amador County, California, and at the Eureka Mill, Plumas County, California. In the third, known as the Grass Valley method, from its introduction and constantly successful use there, no amalgamation is used in the batteries. Blankets and amalga-

mated plates are used, as at the Idaho and Eureka mills in Grass Valley, California, and the former arrangement of the Eureka Mill in Plumas County, California. Fourth, no battery amalgamation with concentration as at the Drumlummon Mill. Fifth, the Boss process.

The example of the treatment of tailings in sluices and round buddles is taken from the Idaho and the Eureka mills, which were formerly on the same vein as the Idaho, and but a few hundred feet from it. It represents the general treatment of Grass Valley. In other mills, such as the Plymouth Consolidated, the Zeile, Amador County, the Providence, the Consolidated Wyoming, the Nevada in Nevada County, and the Sierra Buttes in Butte County, California, and elsewhere, the Frue vanner is extensively and successfully used to concentrate the tails. A number of other concentrating machines, such as the Triumph, have also been introduced in Paradise Mine, Nevada County; the North Star and the Empire mills, Grass Valley, Colorado; Grub Gulch, Frisco County; Sovereign, California; Colonel Sellers and Rio Grand mills, Colorado; and elsewhere.

I. Battery Amalgamation without Concentration.

Examples of the Dakota Mills.

The examples of amalgamation in the battery without concentration are taken from the mills in Dakota, all but one of which, the Caledonia, are under the management of the Homestake Company. These mills are all in Lawrence County, in the Black Hills of Dakota. They are some of the largest mills in the United States. Their names are given below:

Name of the Mill.	Town.	Owners.	No. of Stamps.
Homestake . .	Lead City	Homestake Mining Co.	80
Golden Star . .	,, ,,	,, ,,	120
Highland . .	,, ,,	Highland ,,	120
Deadwood . .	Terraville	Deadwood Terra Mining Co.	80
Golden Terra .	,,	,, ,, ,,	80
Caledonia . .	,,	Caledonia Mining Co.	80
Father de Smet .	Central City	Father de Smet Mining Co.	100
			660

They are excellent examples of the way in which poor ores of low grade can be made to yield a profit by working on a

very large scale. Within an area of about 6000 ft. by 1600 ft. they produced in 1887 $2,271,341 in gold from rock averaging less than $4 to the ton. Their process consists of simply crushing and amalgamating in the battery and on copper plates, the tails not being worth concentration. The ore is free milling. The gold is mostly coarse enough to be caught in the batteries. The ores near the surface are very much oxidised, and produce tails as low as $0.50 to the ton. Those in the lower levels of the mines are less so, and sometimes produce tails of $2.25. When the Homestake Mill was first built, the whole of the ore was not mined, and the yield of the selected ore ran as high as $9.70 per ton. Since then it has been found best to take the whole of the rock between the walls, and the yield has fallen to from $4 to $5, but the immense number of tons milled, and the perfect system of treatment, with the great economy introduced in the use of a very large number of stamps and consequent output, more than compensates for the diminished yield. The company has two mills of its own and manages four others. Its own mills are the Homestake of 80 stamps, in a building 80 ft. by 100 ft., built in 1878 at a cost of $164,500, and the Golden Star of 120 stamps, in a building 120 ft. by 90 ft., built in 1879 at a cost of $251,000. Both of these mills are situated on the mine. They together crush 800 tons of ore a day. The ore from the mine is dumped on to grizzlies. What passes these goes to the ore bins, and what does not goes to the rock-breakers. Of these there are four No. 5 Blake's crushers in the 80-stamp mill and six in the 120. These crushers are 9 in. and 15 in., and are set to crush from $1\frac{1}{2}$ in. to $1\frac{3}{4}$ in. They treat 3.5 tons in twenty-four hours, about half their capacity, which is due to the small size of the opening. The ore bins are made at half pitch. Each pocket is made to serve a battery of five stamps, which are fed automatically, the 80-stamp mill by sixteen Hendy's Challenge self-feeders, and the 120-stamp mill by twenty-four. The arrangement of the mills is such that the ore is not touched by hand from the time it falls on to the grizzlies to the clean-up. Eight men do all the labour required at the batteries for the 200 stamps. The stamps weigh 850 lb., the stems 340 lb., the tappet 130 lb., the head 240 lb. The shoes 140 lb. each. The shoes are of cast iron

and last two months. They are placed back to back with space for the feeders between them, with a discharge each way towards the centre. In the Homestake the distance between the faces of the batteries is 44 ft. In the Golden Star it is 36 ft. The 80-stamp mill makes eighty-four drops of 8½ in. in the order of 1, 4, 2, 5, 3. The 120-stamp mill makes eighty-two of the same height and in the same order. The screens are No. 6 slot. The 80-stamp mill has a 190 horse-power engine with two 16-in. horizontal boilers, and the 120-stamp mill four 300 horse-power boilers of 16 in. in diameter. The engines are placed at the end of the mill between the two lines of batteries. The duty of these stamps, which was at first only 2 tons per stamp, has been raised to 4 tons. This increased output has been made possible by the use of the most improved automatic machinery, reducing the cost of milling in the 80-stamp mill to 90 cents, and in the 120-stamp mill to 71 cents, an excellent example of the greater economy possible in a large than in a small mill. Amalgamation is done in the battery. From 3 lb. to 6 lb. of mercury are added every six hours, according to the richness of the ore. There is a copper plate below the screens on the inside. The battery sluice is 10 ft. long by 4 ft. 6 in. wide. It has four plates; at their end there is a mercury trap in which there are three partitions the width of the lower sluices. The sluice is then narrowed into one 14 ft. long and 2 ft. wide with copper plates. This ends in a grating ¼ in. wide. The pulp runs into another mercury trap and a sluice 50 ft. long, another trap, and then to waste. The clean-up of the sluices is made once a month. The battery is cleaned twice a month. The battery sluice plates are dressed every morning and any amalgam removed. The sluices are cleaned every four days. The power used in the mill is steam; the fuel is wood, which costs delivered $4.75 per cord, or 28 cents per ton of ore crushed. The wood is collected by rail, and delivered to the mills in chutes, which are from 700 ft. to 1500 ft. long. They follow the sides of the hills until they reach the bottom of the valley, where they run flat, and have a movable discharge to deliver the wood to the piles. They are made of 4-in. plank, and are 25 in. wide and 12 in. deep. The inclination is usually 6 in. to the foot. At the bottom they curve flat for about 20 ft. and

then turn up again. This part is movable, and can be turned up or down or to one side at will to suit the place to which the wood is to be discharged. The wood rushes through the trough with great velocity so that a small stream runs on the bottom both to cool it and to serve as a lubricator. The bottom and 9 in. of the sides are lined with $\frac{1}{4}$ in. iron. By changing the vertical or lateral angle the wood may be discharged at any height over a

FORCE EMPLOYED IN 1880.*

Class.	The Homestake 80-Stamp Mill.			Golden Star 120-Stamp Mill.		
	Number Employed.	Length of Shift.	Wages per Shift.	Number Employed.	Length of Shift.	Wages per Shift.
		hours.	$		hours.	$
Amalgamators	5	12	4.00	5	12	4.00
Carpenter .	1	12	5.00			
Day labourers	9	12	3.00	11	12	3.00
Engine-men .	2	12	4.00	2	12	4.00
Feeders . .	2	12	3.50	2	12	3.50
Foremen .	2	12	4.60	2	12	4.60
Machinist .	1	12	4.00	1	12	3.75
Millwright .	...	...	...	1	12	4.00
Watchman .	1	12	3.50	1	12	3.50
Total .	23	...	...	25		

FORCE EMPLOYED IN 1887–8.†

Foreman .	$\frac{2}{3}$	...	6.50	$\frac{2}{3}$	...	6.50
Amalgamators	4	12	3.50	4	12	3.50
Engine-men .	2	12	3 50	2	12	3.50
Firemen .	2	12	3	2	12	3
Crusher-men .	5	10	3	6	10	3
Pipefitter .	$\frac{1}{3}$	10	3 50	$\frac{1}{3}$	10	3.50
Millwright .	1	10	4.25	1	10	4.25
Oilers . .	2	12	3	2	12	3
Feeders . .	2	12	3	4	12	3
Watchman .	$\frac{1}{2}$	12	3	$\frac{1}{2}$	12	3
Total .	$18\frac{1}{2}$	...	...	$22\frac{1}{2}$		

very large area. To prevent the wood falling back a regular and quite a high pile is made in front of the wood pile. The water used is purchased at the rate of $0.50 to $0.57 per stamp per day. Water is so scarce in Dakota that some of the mills under the Homestake management are obliged to run for several months of the year on water which is either pumped from the mines or settled from the mill tails. This settling is done in two catch

* Report of U.S. Census, 1880, vol. xiii., p. 279.

† Geology of the Black Hills of Dakota Rapid City, 1888, p. 95.

pits, one of which collects the coarse and the other the fine tails. The clear water is raised 200 ft. at the rate of 60 cubic feet per minute. When the catch pits are full they are sluiced out. As in the winter, the temperature sometimes falls to 30 deg. or 40 deg. below zero Fahrenheit, the mill water has to be heated. The sluices in the 80-stamp mill are 10 ft. long, 4½ ft. wide, and have an inclination of 2 in. to the foot. In the 120-stamp mill they are 12 ft. long and 4½ ft. wide. There is only a single clean-up pan to each mill. The force employed is given in the Table on the opposite page.

For some time the tails were concentrated, as they contain pyrites, but as these concentrates did not yield more than $5 to $6 to the ton the concentrators were taken out. The percentage of extraction is between 70 and 80 per cent. The gold produced is smelted and assayed at the mill, and leaves the works as bars ready for sale or for the mint. During the year 1883, 69,104.96 oz., from 791 to 821 fine in gold, and 150 to 177 in silver, were taken out. The value of the gold was $1,158,485.85, and of the silver $12,434.15. The fineness of the bullion in gold is 820, and of the silver 170, which is also the fineness at the Father de Smet Mill. The cost of treatment in 1880 is given in the Table below:*

Items.	80-Stamp Mill Treated, in 1880, 153,372 Tons.		120-Stamp Mill Treated, in 1880, 121,910 Tons.	
	Cost per Ton.	Amount Expended.	Cost per Ton.	Amount Expended.
	$	$	$	$
Labour	053.36	81,837.72	033.17	40,433.75
Supplies	010.36	15,887.88	002.41	2,933.75
Water	011.66	17,874.22	010.29	12,548.51
Wood	027.88	42,759.35	024.76	30,186.03
Machinery	016.20	24,847.88	003.26	3,977.20
Candles	000.77	1,185.75	000.13	152.70
Oil	002.42	3,721.00	001.09	1,323.75
Quicksilver	001.79	2,751.71	001.19	1,455.00
Teams	002.15	3,295.00		
Coal	001.79	2,742.20		
Lumber	001.54	2,361.16	001.33	1,631.39
Timber	000.26	397.14	000.07	84.40
Total	130.18	199,661.01	077.70	94,726.48

* Report of U.S Census, 1880, vol. xiii., p. 280.

In 1887 the Homestake produced 96,790 tons, and the Golden Star 146,565 tons, or 243,335 tons for both mills, yielding $903,407.20 of bullion, or $3.71 per ton of rock treated.

The cost of treatment in 1887-8 was :*

Items.	Homestake 80-Stamp Mill. 96,790 Tons Treated.		Golden Star 120-Stamp Mill. 146,565 Tons Treated.	
	Amount Expended.	Cost per Ton.	Amount Expended.	Cost per Ton.
	$	$	$	$
Labour . . .	24,789.90	0.2561	31,838.60	0.2138
Supplies . .	1,263.71	0.0130	1,149.65	0.0079
Water . . .	16,731.37	0.1729	25,097.10	0.1712
Wood . . .	26,773.50	0.2766	40,156.50	0.2739
Machinery . .	8,924.06	0.0922	17,884.96	0.1220
Oil . . .	1,052.69	0.0109	1,233.21	0.0084
Candles . .	156.00	0.0016	214.50	0.0014
Quicksilver . .	1,001.25	0.0703	3,701.05	0.0252
Lumber . .	684.06	0.070	799.16	0.0054
Total . .	81,376.54	0.8406	121,574.73	0.8292

Cost of Mining and Milling a Ton of Ore at the Homestake Mine from June 1, 1887, to June 1, 1888.

	$	$
Mine :		
Labour	1.090080	
Supplies	.054160	
Powder	.011033	
Candles	.016945	
Machinery...	.024601	
Oil	.009639	
Timber	.172628	
Coal	.004100	
	———	1.383186
Deadwork	... ...	.227106
Homestake Shaft :		
Machinery	... ...	.006190
Golden Star Shaft :		
Supplies	.000494	
Machinery...	.010817	
Oil	.000041	
Wood	.016950	
	———	.028302
B. and M. Shaft :		
Supplies	.000462	
Machinery...	.002191	
Oil	.003531	
Wood	.033457	
	———	.039163
Carried forward...		$1.683947

* Geology and Mineral Resources of Dakota Rapid City, 1888, p. 104.

		$
Brought forward		1.683947
80-Stamp Mill :		
Labour	.101867	
Supplies	.005192	
Water	.068752	
Wood	.110018	
Machinery...	.036670	
Oil	.004326	
Candles	.000641	
Quicksilver	.004114	
Lumber	.002810	
		.334389
120-Stamp Mill :		
Labour	.128777	
Supplies	.004724	
Water	.103129	
Wood	.165012	
Machinery...	.073492	
Oil	.005067	
Candles	.003881	
Quicksilver	.015208	
Lumber	.003283	
		.499573
Saw Mill :		
Labour	.024551	
Supplies	.010036	
Machinery...	.002483	
Team	.009583	
Oil	.000329	
Charcoal	.000061	
Candles	.000041	
		.047084
Blacksmith's Shop :		
Supplies	.001986	
Machinery...	.000835	
Wood	.003082	
Oil	.000180	
Coal	.004395	
		.010478
Machine Shop :		
Supplies	.000614	
Machinery...	.000473	
Wood	.007396	
Candles	.000053	
Oil	.000448	
		.008984
Foundry :		
Labour	.038046	
Machinery...	.001002	
Coal	.023716	
Wood	.005397	
Oil	.000085	
Lumber	.000115	
		.068361
Carried forward		$2.652816

		$
Brought forward		2.652816
Tailings Concentrating Works :		
Labour	.001230	
Supplies	.000090	
Machinery...	.000260	
Quicksilver	.000164	
		.001744
Superintendent's Residence :		
Housekeeper	.002465	
Supplies	.003087	
		.005552
Bullion :		
Freight, mint charges, &c.		.039873
Tramway		.044594
Assay office		.002777
Survey		.003241
Interest and discount		.007699
Salaries		.023977
Taxes		.061624
Expense Account :		
General	.014139	
Legal	.012953	
		.027092
		$2.870989

The Father de Smet Mill was built in 1878. A space 200 ft. in length and 80 ft. in width was excavated for the batteries, and this material dumped over the hillside so as to form the front part of the mill. The building is 140 ft. long and 60 ft. wide. From the mudsills to the top of the roof it is 75 ft. It is made in three compartments, in which the different parts of the mill are placed. It is boarded and battened, and the roof is shingled and painted with metallic fire paint. This building has been thoroughly tested, and at times there have been 1,500,000 lb. of ore in the bins without any resulting straining or damage. The advantage of the ore-bin capacity is gained by diminishing the light on the battery aprons as the ore shutes are directly over them, so that the centre of the mill is dark even on a clear day. The Blakes crushers are set 53 ft. above the battery floor. They have been run to their full capacity with the whole mill in operation, without causing much vibration in the building. There are five of them, 15 in. by 9 in. in size, set upon timbers laid longitudinally with the building. They are driven by a 10-in. belt. The mouth of each breaker is set on a level with a raised floor, which is covered with wrought-iron plates, upon which the ore from the grizzlies descends.

It has been proved in the Black Hills and elsewhere that in milling low-grade quartz there is a great economy in large rock-breaker capacity, these breakers being set to crush fine. Each rock-breaker is supplied with ore fed over two grizzlies, which are 4½ ft. wide by 12 ft. long, set at an angle of 42 deg., and extend from the rock-breaker floor to the carway which runs along the top of the mill. Each grizzly is composed of twenty-four bars of iron, 1 in. by 2 in. by 12 in., set 1½ in. apart. The weight of the four grizzlies is 8160 lb.

The ore bins are made in compartments 4 ft. by 4 ft., 10 ft. by 8 ft. There is a funnel-shaped mouth or door arranged for every battery, so as to discharge the proper quantity of ore into the hopper of the Challenge feeder. There are twenty batteries of five stamps arranged in two lines, ten batteries on a side facing each other. They discharge towards the centre of the main building, and are 36 ft. apart. They are built independent of the building, and are braced against each other. Each stamp weighs 850 lb. and drops from 8 in. to 9½ in., making eighty-five drops a minute. The shoes weigh 135 lb., the stem 335 lb., the boss 200 lb., and the tappet 200 lb. The mortars are lined on the sides and ends. The frame and lining weigh 4900 lb. each. From the work of the mill it has been shown that for crushing low-grade ores a square-bottomed die is the best, and that there is no economy in using the dies after their surfaces have become irregular and worn to any extent. In milling gold quartz the principal expense is the consumption of the shoes and dies, and it appears from the results of this mill that it is desirable to cast them as thin as practicable, so as to replace them often, at least once a month.* The mill has twenty Hendy's Challenge feeders, ten on a side. They are at the back of the batteries, the lip of the feeder being on a line with the mortar feed. The screens are 36-wire. Amalgamation is done in the battery; 4 lb. to 5 lb. of mercury are added in a shift.

Each battery has an inclined apron, covered with amalgamated plates. These are visible from all parts of the mill. They are 5 ft. long and 3 ft. 6 in. wide. The plates are in one piece. They incline 20 in. in 10 ft. When they wear they are brazed together, so that they are often in three or four pieces. At

* Trans. Am. Inst. Min. Eng., vol. x., p. 97.

the end of the plates there are five Hungarian riffles across the sluice. At the end of the apron there is a mercury trap with three gates at different levels, the partitions extending to within a few inches of the bottom. These are lined with movable amalgamated plates. The central sluice is copper plated, and is 100 ft. long and 20 in. wide. After the sluice there are mercury traps and the tails then run to waste. They dress the plates every morning and clean up twice a month. The ore runs on an average $3. The cost of mining and milling is about $1.40. The stamps crush 4 tons in twenty-four hours. When the rock is soft they sometimes crush 5 tons. The mill is run by a 155 horse-power engine.

The capacity of the mill was intended to be from 4400 to 5400 tons of quartz a month. Two additional rock-breakers, added since the mill was built, increased its capacity to 6200 tons, and when soft rock is milled, to 7500 tons.

The water for the batteries is supplied from a tank of 20,000 gallons capacity, which is placed on the south side of the main building. This tank contains a large sheet-iron drum, through which the exhaust steam of the engine is passed to heat the water. This is necessary, as the temperature in winter ranges from zero to —35 Fahr.; 2½-in. pipes distribute the water throughout the mill. Between every two sets of batteries for use on the tables there is a water pipe, 1 in. in diameter. The boilers are supplied from a separate tank, holding 15,000 gallons, placed behind the engine-room.

The employés in the mill are: One head amalgamator, four assistant amalgamators, four feeder tenders, two enginemen, one millwright, two firemen, six rock-breaker men, two oilers, one watchman, one carpenter, one blacksmith; twenty-five in all, to run the whole mill at its maximum capacity. They work the same hours and receive the same wages as at the Homestake, except the foreman, who gets $5.

The total production of the Father de Smet in 1885 was $381,697.41 against $474,552.94 in 1884, a falling off of $92,855.53, which is accounted for by a lowering of the grade of the ore from $4.45 to an average of $3.57 per ton. About the same number of tons were milled, 106,855 in 1885, against $106,540 in 1884. In 1886 the mill produced $73,069.

The loss in mercury at this mill during nine months of 1879-80 was only .3 of an ounce per ton of ore stamped.

The cost of construction of the mill was as follows:*

	$		
Mill lumber	13,706.61		
Shingles	378.00		
Masonry	636.75		
Cement	82.50		
Hauling...	705.00		
Planing mill	850.00		
Lime	207.70		
Miscellaneous	367.00		
		$16,933.56	
Whitewashing	359.15		
Copper	1,109.15		
Quicksilver	414.10		
Belting	952.47		
Firearms	68.30		
Telegraph	44.60		
Toll Gate Company	74.55		
		$3,022.32	
Hardware	5,603.27		
Nails	886.33		
Machinery, including lathe	35,566.73		
Freight...	24,075.42		
		$66,131.75	
Total		$86,087.63	
Deduct supplies on hand		13,060.07	
Actual cost of all material and machinery			$73,027.56
Contract on construction	15,000.00		
Labour on pits, tank (48,000 gallons), mill, &c.	4,652.44		
			19,652.44
Total cost of mill construction and material			$92,680.00
Tramway.			
Lumber and shingles	1,528.00		
Labour	1,333.00		
Survey	87.00		
Sundries	188.00		
			3,136.00
Grading Mill Site.			
Labour, materials, tools, &c. ...			4,939.48
Grand total			$100,755.48

The Caledonia Mill was built in 1879, and had sixty stamps, in a building 84 ft. by 50 ft. It has a No. 6 Gates crusher,† which does all the work for eighty stamps with two men. The crusher is worked by an eccentric, and rock of any size that will go into the

* Trans. Inst. Min. Eng., vol. x., p. 99. † See page 411.

hopper is dumped into it from a platform 6 ft. above it. The crusher is sometimes completely covered up with the ore dumped on to it, but still continues crushing so long as ore falls into the hopper. Everything that comes from the mine, large and small, goes through the crusher. It has been running since January, 1888, and recently broke a shaft from a flaw in the steel. There were formerly three No. 5 Blake's crushers for sixty stamps, which kept six men employed. The Gates crusher crushes 260 tons every ten hours, which is all the mill can treat. It now has eighty stamps of 850 lb.; the boss weighs 275 lb., the tappet 125 lb., and the stem 320 lb. The shoe is of cast iron and weighs 140 lb. It lasts three months. Each stamp makes seventy-four strokes of 12 in., and crushes 3.33 tons in twenty-four hours; in 1888, 73,425 tons of ore were crushed. They use wire screens of twenty-five to thirty mesh. Amalgamation is done in the battery, 80 oz. to 100 oz. of mercury a day being added. The battery sluice is 8 ft. long by 4 ft. 6 in. wide. It has one copper plate. This plate narrows and runs on to two blanket sluices, 5 ft. long and 18 in. wide. These blankets are washed with a brush every hour and the concentrates ground in a pan. The battery sluice plates are dressed every morning. The clean-up is made twice a month. The storage bin has 150 tons for every twenty stamps. The yield of the ore in 1887 was $3.52. The cost of mining was $2.45, and of milling, $0.94. $218,684 of bullion was produced; in 1888 the yield was $4.02. The cost of mining was $1.44, and the cost of milling $0.85. The bullion produced was $295,815. What is caught in the riffles averages $15. The bullion is worth $16 to $17, and carries some silver. The rock is very variable in hardness, but everything is crushed. It contained on an average 4 per cent. of sulphurets, which when concentrated runs $90 to the ton. The Gates crusher takes 35 horse-power; the Blakes require four each. The mill has one 42-in. and one 48-in. clean-up pan, and a 72-in. settler; for treating the blanket concentrates for free gold, the power is furnished by two 60 horse-power engines.

The following Table gives the labour employed at the Caledonia mill:

——	Number Employed.	Length of Shift.	Wages.
			$
Foreman . . .	1	...	6.50
Amalgamators . .	2	12	3.75
Enginemen . . .	2	12	3.50
Firemen	2	12	3.25
Crusher-men . . .	1	10	3.00
Millwright . . .	1	12	4.50
Feeders	2	12	3.25
Day labourer . . .	1	10	2.50
Carmen	1	12	2.50
	13		

The Highland Mill was built in 1880, and has 120 stamps arranged as in the Golden Star. In addition, they have a Leavitt stamp of a capacity of 225 tons per day. The stamp building for the 120 stamps is 112 ft. by 92 ft. Three stamps are served by six No. 5 Blake's crushers, and are fed by Tulloch feeders. The distance between the two lines of batteries is 46 ft. The engine is 350 horse-power and has four boilers. The mill produced, in 1887, 146,013 tons of ore, yielding $410,953.54, or $2.82 per ton of rock.

The Deadwood Terra and the Golden Terra both have eighty stamps placed in a line. They were built in 1879 and 1880 respectively. The stamps weigh 850 lb., the boss weighs 250 lb., the stem 350 lb., the tappet 125 lb., and the shoe 125 lb. The stamp makes eighty-seven drops of 9 in. The shoes are of cast iron, and last two months. They use the Tulloch feeder. They have nine to the inch slot screens. They amalgamate in the battery, and add 4 lb. of mercury in twelve hours. The battery sluices are 12 ft. long and 4 ft. 6 in. wide, and end in a mercury trap. They run then into a sluice 8 ft. long and 1 ft. 2 in. wide, copper plated. These all end in one running 50 ft. from each side of the mill, and at right angles to this an 8-ft. sluice, then into one 125 ft. long and 1 ft. 6 in. wide. They clean up twice a month. The battery plates are dressed once a day. The short sluices are cleaned three times a week. They crush about 4 tons per stamp. The mill is run by a 155 horse-power engine. The product of these mills, with the Father de Smet, in 1887, was $639,229 from 216,361 tons of ore, or $2.02 per ton.

II. Battery Amalgamation and Concentration.

A. Hendy's concentrator.

B. The Frue vanner.

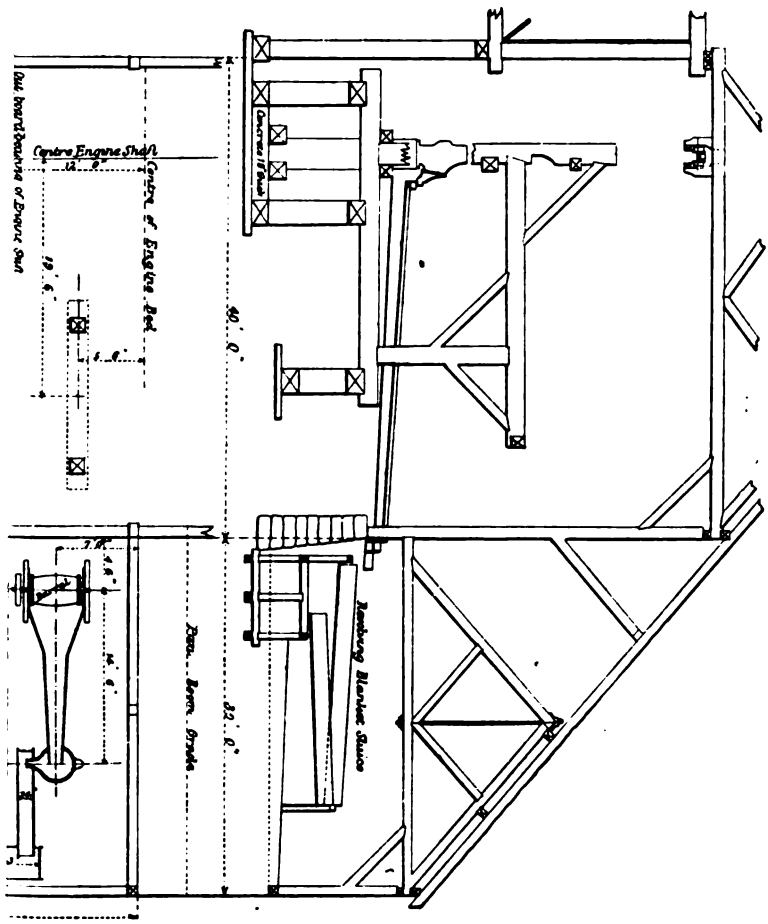
C. The Triumph concentrator.

A Hendy Concentrator.

The mills using battery amalgamation and concentration will be classified according to the concentrators employed. Those in most general use are the Frue, the Triumph, and the Hendy. Other concentrators, such as buddles, have been extensively used; they will be treated in the description of the mills where they are used.

The Plumas Eureka Mill, of Plumas County, California, as constructed previous to 1874, is an excellent example of a low-grade mill. Fig. 52 shows a section of the mill on the line of the stamps and revolving blankets and the ground plan with the position of all the appliances used. The mill had then forty stamps of 780 lb. making eighty to eighty-five drops of 8½ in. per minute. It had a capacity of 60 tons in twenty-four hours. Owing to the altitude of the mine, which is about 7000 ft., and the depth of the snow in the winter, they cannot work during the winter season. The mill was run by water power, which was supplemented by steam when the water gave out. Fig. 52 shows the mill as it was originally constructed with only forty stamps. From the battery the ore discharges on to copper-plate sluices, of which there were two for each battery, which were used alternately. Those for ten stamps are placed together, so that there were four sluices between each two sets of batteries, or sixteen in all. These sluices were 8 ft. long, and they were continued into 20 ft. of fixed blanket sluices. The tails from one set of four sluices ran over revolving blankets, which have since been abandoned. They were arranged to work automatically and discharge their contents into a vat underneath them. It had a 100 horse-power engine and used both water and steam. The mill cost $100,000. In 1880, with eighty stamps, 36,735 tons of ore were milled at a cost of mining of $2.65½, and of milling of $0.905.

As arranged in 1874, the Plumas Eureka Mill, Fig. 53, had twenty-two Hendy's concentrators, two Patton pans, two settlers and a clean-up barrel, and forty stamps. The ore averaged about



A, AS ARRANGED PREVIOUS TO 1874.

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$15, and cost $4.50 to mine and less than a dollar to mill. In 1883 the mill yielded $35,000 per month. The cost of mining was $2.655 and of milling $0.495. In 1884 it cost $3.52 to mine and 49 cents to mill, but the yield of the ore had fallen to $6.72. The quantity of sulphurets was 1 per cent. In 1888 the ore yielded $7.93 per ton. The mill was rebuilt in 1880, and is now driven by a Knight's wheel 8 ft. in diameter. The water is brought from a mountain lake four miles distant, and is delivered at a pressure of 260 ft. It has a 75 horse-power engine to be used when the water gives out. It has sixty stamps of 850 lb. each, and made eighty to eighty-five drops of 8½ in. per minute with a capacity of 2.55 tons per stamp, or 153 tons in twenty-four hours. The shoes and dies are of steel. They wear 1.06 lb. to the ton of ore crushed. The cost of milling is 49½ cents. The cost of mining $3.20. The ore is free milling, but contains 1¼ per cent. of sulphurets, which are worth $30 to the ton. These were formerly treated in Patton pans. They are now treated by chlorination at the company's works and cost $10 per ton. The mill now has three Blake's crushers, twelve Challenge feeders, two Patton pans, a clean-up barrel, a batea, twenty Hendy concentrators, three Duncan concentrators, five Patton concentrators, and a "plate house" for the treatment of the tails.

The battery screens are No. 8 diagonal punched slots. The lower rail of the screen frame is made 8 in. wide when the shoes and dies are new; when the shoes and dies are worn, this is reduced to 6 in. in order to keep the same height of discharge. The discharge surface is 45 in. long and 6 in. high. Mercury is used in the battery. The inside of the front of the mortar is lined with copper plates, which are secured to wooden pieces made to fit the metal tightly. These plates for the step of the mortar are 51 in. long. They are made at an angle to correspond with the vertical part of the die seat and the inclined step of the mortar, each part being 5½ in. long. The sides of the mortar are also lined. Duplicates of these plates are always kept, so that there may be no delay in changing them when they are to be cleaned. The amalgam collects on the surface of these plates in irregular bunches and ridges, the position of which is determined by the swash of the battery produced by the order of

fall of the stamps. It is evident that with a given ore there is an order of fall which will correspond with a maximum amount collected, a point which should be carefully studied. The mercury is introduced through the charging slot about once in two hours. Sometimes the interval is as short as every half-hour if the work is not being properly done. The quantity depends upon the richness of the ore. The effort is made to distribute it as evenly as possible under the stamp. To get the best results from the amalgamation in the battery, it is necessary that the ore should be stamped as fine as possible without making float gold. The size of the screens is therefore to be determined by experiments on the ore. The movement of the pulp in the mortar is such that there are periods of rest favourable to the deposition of the gold, followed by periods of motion which discharge the pulp. The coarse particles of gold will be too heavy to be carried by the swash through the discharge, and they will consequently tend to remain on the mortar bed, where, as the periods of rest are comparatively long, they will be collected by any mercury or amalgam with which they may come in contact. That this is so is shown by the fact that the quicksilver escaping from the battery contains very little gold. Most of the gold caught after the pulp leaves the battery has not been touched by the battery quicksilver. A small portion of the mercury is broken up by the blow of the stamp, which, with the gold it amalgamates on its way out of the mortar, catches more gold, and most of this lodges on the copper plates on the lip of the mortar, or on the apron on the outside of the battery. This amalgam should be easily, but not too easily, indented with the finger.

Fig. 53, which is the plan of the Plumas Eureka Mill, up to about 1880, shows the arrangement of the different discharges.

For the upper set:

Battery	1	discharges	into	concentrators	1	and 2
,,	2	,,	,,		3	,, 4
,,	3	,,	,,		5	,, 6
,,	4	,,	,,		7	,, 8
,,	5	,,	,,		9	,, 10
,,	6	,,	,,		11	,, 12
,,	7	,,	,,		13	,, 14
,,	8	,,	,,		15	,, 16

For the lower set:

Concentrators	1 to 4	discharge into	concentrator	17
,,	5 ,, 8	,,	,,	18
,,	9 ,, 12	,,	,,	19
,,	13 ,, 16	,,	,,	20

From the trough the pulp runs on to Hendy's concentrators, of which there are three rows at different levels. The first row consists of sixteen in two rows of eight, or two to each battery; the second row of four, or one to two batteries, and the third row of two. This last row catches any amalgam and sulphurets that may go over. In the first two Hendy's concentrators, amalgam, free mercury, and sulphurets are caught. They discharge constantly, except when those below them are full, and require cleaning from pulp; the lower ones discharge constantly. What discharges from them goes to the tail sluice. The amalgam and pyrites discharge into the last two concentrators; these are only discharged when they are full. The discharge from them goes to a rocking sieve attached to a pan which is supplied with water. This pan is 18 in. long, 12 in. wide, and 4 in. deep. The coarse particles which escape, when the battery is open, are caught here. They discharge into a box 2 ft. wide and 2 ft. deep, and consist of sulphurets, which are dried and sent to the chlorination works. They assay from $100 to $50 in gold and silver to the ton. The silver, which is sometimes as much as $10, was entirely lost in the chlorination works. From the four concentrators the pulp is shovelled out of the box in front into a box 3 ft. 6 in. long, 2 ft. 6 in. wide, and 10 in. deep, inclined 5 in. in its length, into which water constantly runs. From this box the concentrators run into a vertical pipe 3 ft. 6 in. high through an inch nozzle, in the bottom of which 1 lb. of mercury is placed connecting with a box, on the outside of which is a vertical trough 6 in. by 9 in., with a cover working in a groove, and 3 in. holes 2 ft. from the bottom, and 3 in. apart. In the bottom of the mercury pipe is a plug. The pipe dips into a trough containing water. This trough is 2 ft. 6 in. long, 2 ft. wide, and 16 in. deep. An inch pipe with a head of 12 ft. to 15 ft. discharges water into the bottom of the mercury pipe. This forces the sulphurets through the discharge holes, and gives the

amalgam and mercury time to settle at the bottom. This is cleaned up with a rubber and sheet-iron shovel every night, and about $25 a day are extracted from them. The sulphurets discharged from the holes run into a trough 12 ft. long, 13 in. wide at the upper end and 7 in. at the lower, and 2 in. deep, with amalgamated plates, and is there discharged into the set of two concentrators. The copper plates are 30 in. long, and are juxtaposed. From the last two concentrators the sulphurets go to chlorination.

The gold caught from the last two troughs and from the tank is caught here; 1¼ tons of pulp pass through it. The ore caught is quite poor. Every three days the amalgam caught in the high box is cleaned in the barrel, Q, Fig. 53, which revolves twenty times per minute.

In the present Plumas Eureka Mill the material discharged from the first set of sixteen concentrators runs from each eight into two concentrators, and is then treated in two Patton pans and two settlers. The settlers discharge into two concentrators.

A general clean-up of the mill is made at intervals. The time between these clean-ups is called a "run." A run will vary according to circumstances from twenty to sixty days. For the purpose of making a general clean-up there are two methods, stopping the whole mill, or what is better, as it saves time and does not diminish the production of the mill, cleaning each battery separately. When this is done the mortar in every battery is cleaned once in two weeks. The screens are then removed, the shoes and dies and boss taken out, and any amalgam found in the shoes or dies is carefully picked out. Duplicates of the battery plates are kept on hand, so that to clean the mortar only one battery is hung up for ten minutes at a time, the plates taken out or reamalgamated and put into the next battery. It takes two men five hours to take out the plates, clean, and replace them in the next battery. Sixty per cent. of all the gold is taken from these plates. At the end of the sluices there are various kinds of mercury traps which have several partitions. These sluices are cleaned up more or less frequently, generally once a month. The battery is cleaned up generally twice a month. The apron plates every four to five days.

All the amalgam from the battery sluices and plates is collected together, and is diluted with a large quantity of mercury, generally one to one and a half flasks. This amalgam contains about 60 to 65 per cent. of quicksilver and is distilled.

Every other day all the amalgam from every part of the mill is collected and cleaned in a barrel Q, Fig. 53, which at the Plumas Eureka Mill is 3 ft. in diameter and 4 ft. long. At the Keystone Mill it is 5 ft. long and 2 ft. 6 in. in diameter. It revolves twenty times a minute. The charge is 700 lb. of amalgam and 20 lb. of mercury, or more if the amalgam is very rich. Twelve battery shoe shanks are put into the barrel, which is then filled with water, so that it will be three-fourths full. It is revolved for twelve hours, and the charge taken out. There is always a considerable amount of sulphurets charged in and with the amalgam, from the clean-up of the last sluices. The barrel is washed out with water. The wash-water is run over a sluice with amalgamated copper plates, which is 14 ft. long and 14 in. wide, connecting with a batea, Fig. 52, which is an iron pan 20 in. in diameter and 2½ in. deep in the centre, suspended by three hooks 5 ft. long. A backward and forward motion is given to this, so that it works like a pan in gold washing.

When in working a new battery or new ores a thorough knowledge of the working of the batteries, the peculiarities of the ores and their values, has been acquired, it can be very closely determined by the weight of amalgam cleaned up each day, what the full gold resultant value of the entire run will be by knowing the proportion which the amalgam making on the outside plates bears to that on the inside of batteries. This proportion should be as one for that outside and three for the battery plates and aprons. In the final clean-up the plates must not be scraped or skinned too close—a thin film must be left behind, it will assist amalgamation when used again. All the amalgam runs into the batea, and the lighter particles are washed over into the sluices in front of it, which are 10 ft. long, and end in a tank 7 ft. long, 3 ft. wide, and 3 ft. deep.

Amalgam as it is taken from the batteries and outside plates, concentrators, grinding pans or settlers, whether gold or silver

is seldom sufficiently clean for retorting, inasmuch as it will be found to be intermixed with small quartz pebbles, pyrites of iron, sulphurets and chlorides, and other imperfectly reduced metals and minerals, and chips and bits of iron and other foreign substances. Such amalgam must be treated in a bath of quicksilver, in which the whole mass must be stirred until all foreign substances are brought to the surface and then skimmed off with a woollen rag or sponge, wet with a weak solution of cyanide of potassium, and then washed in a miner's pan, and all particles of iron or black sand extracted by the use of a magnet. It is then carefully placed in another vessel for further treatment. The mortar cleanings and valuable refuse which accumulates should be subsequently ground and treated in a clean-up pan or revolving barrel and batea. It takes one and a half hours to clean the amalgam after it is placed on the batea, when there is a general clean-up. It takes about half an hour when the cleaning is done every three days. The resulting amalgam is then squeezed through a chamois skin or canvas bag, ready for the retort.

After leaving the concentrators at the Plumas Eureka, the tails pass into the "plate house," where there are thirty-six sluices 5 ft. long and 15 in. wide covered with silver-plated copper plates; the tails are turned on to these plates; clear water is run on each sluice. The plates are soon worn and require replating, which is done in the mill. The yield of the plate house is about $300 per month, which a little more than pays expenses.

The loss of quicksilver in the mill per ton of ore is 0.27 oz. The gold at the Plumas Eureka varies in fineness, that from the batteries and first copper plates is 700 fine, while that from the pans varies from 500 to .018 in fineness. The amalgam yields 20 per cent. of its weight in gold.

This mill formerly used blanket sluices and the Patton pan and concentrators. These are now thrown out for Hendy's concentrators. The tailings in 1888 assayed $1.50 to $2 a ton as a maximum, when formerly they ran from $4 to $7 above the buddle, and $5 below it. The Paine and Stevens buddle was formerly used here, but its use has been discontinued. During the first year that there were twenty-two Hendy's concentrators

in the mill, the repairs did not amount to $5 a concentrator, and were all to the Babbitt metal boxes of the vertical shaft. The concentrators have to be stopped occasionally to be cleaned from the gum in the boxes. This should be done every seven or eight weeks, but it is not done oftener than once in three months. If this was carefully attended to the probability is that there would be hardly any repairs to the concentrators.

The discharge from all the concentrators runs into a sluice composed of four sluices side by side, 50 ft. long, 10 in. wide, and 16 in. high. Across these, riffle bars ½ in. wide are placed every 12 ft. One of these is placed in position every two hours, until the whole trough is full to the top. Two of the four sluices are worked at a time. When two are full the tailings are turned on to the other two; clean water is then turned on to the full ones and they are washed with a broom, taking the riffles out in the same order that they were put in. One man working ten hours a day can collect 75 lb. of sulphurets, which contain about $150 to the ton. Twelve men do all the work of the 60-stamp mill.

The sulphurets caught in cleaning the amalgam are put through the concentrators. At the Plumas Eureka Mill these are roasted in a Brückner's furnace and worked in pans. In some mills they are mixed with salt and allowed to weather for a year before they are treated in the pans.

The mine of the Keystone Mill, Amador City, Amador County, California, has been worked for thirty-five years. It is situated directly behind it. The ore is discharged into the stamp shoots. It is quartz with about 1½ per cent. of pyrites containing some antimonial and arsenical sulphurets and a small quantity of free gold. The concentrates assay from $80 to $135 per ton, of which 83 per cent. by weight is gold and 17 per cent. silver. It assays about $14 in free gold to the ton. The mill was formerly continuously run by a 96 horse-power engine, which had two boilers, burning 8½ cords of wood per day; when there is sufficient water it is now run by a 7-ft. Knight's wheel, with 254 ft. pressure. It is wet crushing and uses 125 in. of water; for mining and milling together 180 in. of water are used; the water was formerly supplied by a 6-in. pipe with a pressure of 50 ft. It is heated in a tank

by coils of pipe through which the waste steam discharges. The mill has forty stamps in eight batteries. They weigh 750 lb. each. In 1874 they made eighty-five drops of 8½ in. per minute, and crushed a little less than 2 tons to the stamp. They now make ninety-two drops of 6 in. to 7 in. to 8½ in., and crush 2½ tons per stamp through a No. 8 straight slot. The batteries are both high and low discharge. The roller self-feeder is used.

The order of drop of the stamps is 1, 2, 3, 5, 4, in one battery, and the reverse in the next. The stem weighs 310 lb., the tappet 85 lb., the boss 215 lb., the shoes 120 lb. The shoes and dies are of iron They cost only 2 cents per ton of ore stamped. Chrome steel was used for a time, but the shoes and dies lasted only sixty days, and cost more than the iron. The apron plates of the mortar are 48 in. by 13 in. Two batteries discharge on to an apron 12 in. wide and 5 ft. long. This ends in a sluice 12 in. wide and 10 ft. long, set at an incline of 3½ in. in 10 ft. The aprons are lined with copper, except for the last foot, where they run into a pointed box 1 ft. deep and 1 ft. wide at the top. On the side, near the bottom, a ½-in. pipe discharges water, which rises and overflows at the top. The box sets over a tank. In the bottom there is a hole, closed by a wooden plug, which rises above the level of the water. The mercury and amalgam collect here. From the pointed box the pulp runs on to two sets of Hendy's concentrators on one level, and one set on the level below, making twenty-eight Hendys in the mill. Nine men do the work of the mill.

Seventy-five per cent. of the gold collected is caught in the battery, and twenty-five on the plates. When the amalgam from the batteries has collected to the amount of 800 lb. to 1000 lb., it is cleaned in an iron barrel 4 ft. long and 2 ft. in diameter. The front of the man-hole is a splash-box 12 in. long and 2 ft. wide, closed by a slide. This ends in a spout 30 in. long and 13 in. wide, at the lower end of which there is a screen with ⅛-in. holes, which allows what passes to fall on a batea which is 36 in. by 30 in., moved by an eccentric crank. From here the tails flow to a launder covered with plates 60 in. by 12 in. From here, with a turn nearly at right angles, it flows on to a sluice 32 in. long and from there to a box 55 in. by 30 in., and 30 in. deep. The battery sand is also treated in this barrel with fresh mercury

with broken stems or any other old iron, and made to revolve for four hours with forty revolutions a minute. Most of the mercury and amalgam is caught on the batea and on the amalgamated plates; the rest is caught in the box below. This box has a partition in it, 1 ft. from one side, so as to prevent the mercury from being carried over.

The ordinary sluice plate amalgam is not put into the barrel, but is cleaned in an iron mortar by itself. There is caught on the bottom of the Hendy a hard dry amalgam, which is cleaned in the batea. It collects on the bottom of the iron pan, and has to be scraped from it.

The tails from the Hendy's concentrators run into a building outside the mill, where they are treated on sluices called "strakes." There are two sets of four of these sluices, with eight boxes, each 16 ft. long, or 128 ft. for the eight lengths. The riffles are ½ in. high and are put in at the end of every box, and are held down by a piece of ¾-in. round iron slanted up stream. In each of these boxes one riffle is put every hour. Over the head of the sluice is a head-box for clear water, with an opening for each sluice. When the sluice is full the riffles are taken out, taking out two in the first four, and turning over the concentrates walking up stream, clear water only being allowed to run. In the next four only one riffle is taken out, and so on. Only eight riffles are used in the sluice. This takes two men constantly at work. These tails belong to the mill. What overflows from the tails runs over forty-nine sluices 20 in. wide and 20 ft. long slightly inclined, with a supply sluice and a head box for clear water. These sluices are covered on the bottom with duck 22 in. wide, made to turn up 1 in. on the side of the sluice and tacked down. The pulp runs over these for four hours. They are then cleaned up, turning on clear water. At the bottom of the whole line is a catch-box, by the side of the discharge sluice. An apron is put over this to continue the sluice to the catch-box, and it is swept down with an ordinary broom, first down and then up, and the pulp turned on again when the apron is dropped. It takes but a few minutes to do this washing. One man does all the work. They are swept four times a day. The concentrates so collected amount to four tons in a month.

At the Spring Valley Mine, about two miles from the Keystone, the pulp from the Frues runs into sluices, with an incline of 1 in. in 12 ft., which are riffled like those at the Keystone. There are six sluices 60 ft. long. At the end of these there are thirty duck sluices, or "strakes," 20 in. wide and 16 ft. long. This work is let out on contract, and pays about day's wages.

In the first five months of 1888,* the value of sulphurets caught in the Keystone Mill was as follows:

	$
Caught on the Hendy's concentrators	93.33
,, riffled sluices	71.52
,, canvas strakes	44.00
Upper tails leaving the mill	0.70
Lower tails below the sluices	0.50

The loss in mercury is 0.486 oz. per ton of ore. The cost of milling per ton of ore is $0.75, as follows:

	$
Labour	0.27
Water	0.19
Shoes and dies	0.12
Screens	0.01
Mercury	0.02
Sundries	0.14
Total	$0.75

Charging mercury into the stamp battery should be regulated by the consistency of the amalgam on the outside plate near the discharge; if it feels harsh and crumbly to the touch, it requires more; if thin and slippery, less; it should feel about like moist brown Muscovado sugar. It is the usual custom to charge the battery every hour for ordinary rock, and for rich rock every half-hour. A mineral containing one ounce per ton will require the contents of a wooden mustard spoon put in every hour.

It is generally about an ounce of mercury for every ounce which can be got out of the ore. If the gold is very fine more is needed. When the ore is rich, assaying $20 to $30 per ton, the amount of mercury added in the battery will be about three thimblefuls. This quantity is added every half-hour; when the ore is poor, that is to say, yielding about $10 per ton, a thimbleful is added every hour. In some mills 4½ in. of a stick of sodium about ½ in. square is added to every flask of mercury which is used in the battery to make the mercury lively. Exactly

* Eighth Report of the California Mining Bureau, page 64.

the right quantity of mercury must be found by experiment. Too much dilutes the amalgam, and makes it so liquid as to allow of its being carried off, and if the excess is great the losses of the mercury and gold will be maxima. If there is too little, less gold will be caught, and the tailings will be rich. It has been found by experiment that with some ores, adding small quantities frequently does not give any better result than adding a large quantity once in two or three hours. With other ores, as here, they are better treated by adding small quantities at a time. The quantity to be used is judged of by the appearance and consistency of the amalgam. Inside the mortar it should be too hard to be readily pressed by the fingers, and yet not dry enough to be broken up by the stamp into particles fine enough to pass the screens. If the amalgam is soft no more mercury must be added until it becomes hard. If it is too dry some must be added. The blow of the stamp divides it into small globules and flours some of it. The motion of the pulp caused by the blow of the stamp brings the fine particles in contact with the fine particles of gold, and amalgamates some of them, and what are not caught in the mortar and on the mortar plates are carried into the discharge. The coarse gold, however, will generally be too heavy to be raised to the level of the discharge, and it is either caught by the mercury and amalgamated, in which case it generally attaches itself to the copper plates or sinks to the bottom of the mortar, and is caught there. Two-thirds of all the gold is taken from the plates in the battery, and most of this from the mortar step. If it is not caught in either of these ways in the mortar, it is thrown out from under the stamp until it is either beaten into leaf or broken up to such a size, and has its gravity so reduced by the mercury, that it can be moved by the water, and carried out by the mortar.

For rich ores the battery water is used in sufficient quantity to prevent the pulp from catching on the aprons, or about thirty times as much water as rock. For poor rock the quantity must be increased to about thirty-five times. The supply of water to the mortar should be carefully regulated. When there is too little, the plates become covered with pulp so that they cannot act, and very rich amalgam under such circumstances is likely to

become pulverised and carried off. The mortar fills, so that the screens do not do their whole duty. When there is too much the crushing is coarser, and the pulp remains too short a time in the mortar, and is rapidly carried over the sluices. The addition of a greater quantity of mercury in such a case does not help the matter; for though more amalgam is formed, from its being too liquid and of less gravity, it is more easily thrown out of the mortar. The water should be as warm as possible without being too warm, for when the temperature is too high the amalgam is too liquid, and does not adhere to the plates. When the water is at the right temperature, the amalgam adheres to the copper plates, which become rough with it, and thus help to catch the other particles of gold. When the water is too hot the plates are more likely to remain smooth, and to allow the mercury to slip over.

Directly in front of the apron of the mortar there is a splash-box 18 in. high, made of thin stuff, which can be easily moved. This box has three discharge openings which can be closed with wooden plugs. The pulp from two batteries discharges into a box 4 ft. 6 in. long, 1 ft. wide, and 10 in. deep, in the middle of which, and raised above it, is one 2 ft. 6 in. long, 1 ft. wide, and 3 ft. deep, which serves to hold the plugs and pails used in cleaning up the aprons and the mortars. From the ends of the large box the pulp is discharged into sluices, Fig. 53, which connect with Hendy's concentrators. These sluices are 20 ft. long, 15 in. wide, and 3 in. high, with an incline of 1 in. to the foot, or about 4 deg. The bottom is covered with amalgamated plates, which are 4 ft. long. They are made perfectly true, juxtaposed and fastened by copper screws. Near the end there is a drop-box 14 in. wide at the top, 6 in. at the bottom, and 14 in. high, which catches a large part of the mercury which discharges from the battery. The amalgam, however, goes beyond this point. At its end the sluice is divided in two of half the size, to deliver the ore from one battery on to two concentrators.

The battery pulp runs over these inclined sluices, or through machines, with a movement which is very nearly uniform, and which is not therefore favourable to the deposit of the heavier particles, to provide for which the pointed drop-box or pockets

in the sluices are made. As it runs over the sluices in a thin layer, the larger pieces of gold and amalgam slide slowly along on the bottom, and are caught by the copper plates, the drops, or in the concentrator. The float gold, however, and the small pieces are carried near the top, and have no opportunity to adhere to the plates, which is the reason why so much of the fine particles of gold and floured quicksilver are carried off, so that the tailings, after having passed over all the appliances for catching the gold, still contain a considerable amount of it.

The difficulty of treating by amalgamated plates arises from the ease with which the surface becomes oxidised, owing either to the formation of the oxides of copper or of other metals, or to the adhering of foreign substances, such as grease, to it. The difficulty of preventing the formation of these oxides is much greater than it would at first appear. The mill water is likely to contain many substances which would tend to form them, and which are either contained in the water before reaching the mill or introduced into the water afterwards, or taken from the ore while in contact with it. Carbonic acid discolours the plates, and iron and copper sulphates will do the same. This is prevented by the addition of a little lime to the water, which decomposes the sulphates, and by constantly rubbing the plates with sal-ammoniac, or other substances which will dissolve the oxides and grease. It is generally quite impossible to entirely prevent this action, and hence it is necessary to constantly watch and clean the plates, and to take every precaution to see that the water used in the mill is pure.

Alloys of copper and other metals have been used for making the plates, so as to keep them clean, but these have generally been found to cost more, since the alloy makes the copper brittle and renders it very liable to break. Silver and gold-plated plates have been used with success, so far as the amalgamation is concerned, but gold-plated plates are very expensive. These plates are, however, very sensitive, and do not tarnish as quickly as ordinary amalgamated copper plates.

There are several methods of preparing the plates, which are used indifferently for the battery or sluices. The first is the ordinary copper plate, to which mercury is applied after it has

been carefully cleaned. These do not catch the gold very well at first, and do not act quickly until they have become thoroughly coated with gold amalgam, and in order to make them efficient from the start they are usually coated with gold amalgam sufficiently thick to insure that the copper will not be acted upon. Such plates will act until they are worn out, and the gold then recovered from them will pay for new ones. The second is treating the plates in the usual way and then coating them with silver amalgam. These plates are efficient from the start, as the silver amalgam is little by little replaced by gold amalgam. The plate is good until it is worn out. It is preferred in some places, as it is cheaper to prepare, and in the end arrives at exactly the same purpose as the gold amalgamated plate. The third is silver-plating the copper plates. These act quickly at first, and require but little care and attention until the plating wears off, when they must be replated. Such plates are in general use. They hold a very large amount of quicksilver, and soon pay for themselves. They are made plain, corrugated or riffled, of any size or thickness desired. The plain ones answer best. When the mill is near a large city, it is cheapest to have the replating done at some establishment which makes a business of plating. When it is not it must be done in the mill. The silver is paid for by weight. A square foot of copper plate requires from 1½ oz. to 3 oz. of silver, and costs from $1.50 to $6, according to the weight of the silver put upon it. Upon the plates prepared in any of these ways the gold collects, not in layers of uniform thickness, but in irregular patches, bunches, and ridges, the position of which is dependent upon the way the swash produced by the motion of the stamps flows over it.

Only the very best qualities of copper that the market affords should be used for making amalgamated plates. It is especially desirable that the copper should not have been rolled to such an extent as to make the surface bright and hard. It will then not be sufficiently porous to absorb the mercury. If the copper has been rolled, it is necessary that it should be annealed before using. This is generally done by heating the plates from the underside so that wood on the upper side will take fire from the heat of the plate. The plate must be heated as uniformly as possible, and care must be taken that the upper surface does not

oxidise during the operation. It must then be straightened by pounding it with a wooden mallet, the surface of the plate being protected by a piece of hard wood. The plate must not be struck directly, for this would be likely to batter the surface. By such treatment copper, which is compact, is rendered porous, and capable of absorbing the mercury. The best plates for amalgamation should be very smooth and soft on the upper surface. The outside plates should never be less than $\frac{1}{8}$ in. thick, and inside $\frac{3}{8}$ in. The least weight that it has been found practicable to use for the plates is about 3 lb. to the square foot. Within certain limits the heavier the plates are the better. The amalgamated plates are fastened to a table with copper nails or iron wood screws, whose heads must be flush with the surface of the plate. It is much better practice to fasten them by bolts, or by cleats, clamps, or even wedges on the sides of the sluice. Brass screws should not be used, as they become quickly amalgamated, and break off. As soon as they are fastened, the surface of the plates is dressed, and made as smooth as possible with a mallet and block; they are then scoured with sand, Bristol brickdust, or finely sifted oak wood ashes, and rubbed with fine emery paper, so as to make them perfectly bright; or they may be washed with strong soda alone, which brightens them and removes all traces of grease. They should be rapidly rubbed bright and clean with a woollen rag dipped in a solution of one part nitric acid to nine of water and thoroughly washed off with clear water when clean, and rubbed over with a solution of cyanide of potassium, using half an ounce of cyanide to a pint of water. The plate is then carefully washed with warm water, as any extra cyanide, if left for any length of time on the plate, would tend to make the copper dull. The plate so prepared is clean and bright, and ready to absorb the mercury. It is then rubbed with a mixture of fine mill tailings, or any other fine sand, and powdered sal-ammoniac, which contains a small amount of mercury. The object of the sal-ammoniac is to keep the surface bright, to dissolve away any commencement of oxidation, which begins very shortly after the plates are exposed to the air, and which, even in very small quantities, would hinder the amalgamation. This mixture of sand and sal-ammoniac is rubbed on with a brush, and while this is being done, mercury is

sprinkled over the plate until it has absorbed as much as it can take up. The mixture is allowed to remain on the plate for half an hour; it is then again washed with water and cyanide of potassium, and more mercury is added if the plate will take it up, until the entire plate is thoroughly coated and is bright and soft to the touch. Gold amalgam is then rubbed over the plate; or if for any reason it is desirable not to use it silver amalgam will answer the purpose. In either case it is rubbed in with a piece of india-rubber belting, using a solution of sal-ammoniac to keep the plate bright. The whole object of this preparation is to hasten the formation of the gold amalgam, which would take a much longer time in the battery if it were left to itself, and would not then be so effectually done as if the plate, when first put into the battery, had the amalgam already upon it.

New mercury, never before used, should be prepared for the plate. The flask is emptied into a shallow iron dish; if wet, it must be dried with blotting paper. To a quart of dry mercury five small chips of metallic sodium, about the size of those cut from lead pencil in repointing it, are added. They should be dropped into it as cut from the sodium ingot. These pieces will move about on the surface of the mercury when they are clean, when somewhat tarnished they may need to be pushed under with a dry stick to cause them to combine with it. When the sodium is all dissolved and quiet, the mercury is stirred a few times with the point of a new ten or eightpenny cut nail; if on examination of the nail, the four corners are found to be slightly amalgamated, the quantity of sodium is correct; but if too much sodium amalgam has been used, clean mercury will have to be added to weaken it, until another nail shows only the four corners amalgamated and no more. To the prepared mercury 5 oz. of silver, either in filings or dissolved in acid and precipitated by copper or by zinc, is added and thoroughly stirred in until it is dissolved in the mercury; this is facilitated by slightly heating it. It is now ready to apply to the plates. Squeezed or distilled mercury contains enough fine gold, even after retorting, to need only the addition of metallic sodium to make it equivalent to prepared mercury.

Prepared mercury is suitable to commence feeding the battery with until a supply of squeezed mercury is obtained; the latter must, however, be kept charged with sodium up to the nail test at all times, as is also the case with retorted mercury, which carries over enough gold in distilling to make the mercury sensitive in connection with the sodium, and prevent sickening and flouring in presence of sulphurets. The reserve of prepared mercury should always be kept covered with a thin layer of cyanide of potassium solution.

The quicksilver used should always be free from the base metals. It makes no difference, however, how much gold or silver it contains. The reason why the base metals cannot be used is that they oxidise very readily. The different salts of mercury, salt, sulphuric and muriatic acids, and sodium amalgam have all been used, but are not always accessible.

The amalgam on the plates should be carefully watched. If it is too soft, from an excess of mercury, it will drain from it and flour. If it is too hard it will not catch the gold at all. It is easy to ascertain its quality for the maximum amount of gold which can be caught. A yellow stain frequently comes on copper plates, which can be removed with potassium cyanide, which must be used with care, however, as the over use of it causes the amalgam to soften. In some of the mills copper plated sluices are made to drop 5 in. or 6 in. every few feet, so as to better turn the pulp over.

The most careful attention must be given to the plates. If any film forms on them they must be cleaned at once, for the amalgamation becomes less active with the commencement of the formation of a film, and if the latter is allowed to go on may cease altogether. Sal-ammoniac and potassium cyanide are the great remedies for this. The reason why the plates are washed so often with clean water, and then with ammonium chloride, is to dissolve the oxides. The potassium cyanide is used to brighten the plates. Generally the chloride is allowed to remain a few moments on the plates; but if the cyanide was allowed to remain any length of time it would deaden the lustre which it has given. The amount of mercury should always be as much as the plate will hold without running off.

If the ore is very rich, the plates inside of the battery should be changed and cleaned up as soon as the amalgam lays up in ridges, so that it may not be broken down by the action of stamping. In crushing medium or low-grade rock this necessity will seldom occur. The first 3 ft. or 4 ft. of the outside plate should be scraped with a piece of old rubber belting, the edges of which must be kept sharp, every twelve or twenty-four hours, according to the quality of the pulp treated; for 1-oz. to 2-oz. rock, when worked continuously, once in twenty-four hours will answer. For inside plates twice a week is sufficient, unless over 2-oz. rock is going through.

Swinging silver-plated amalgamated plates are now used in many of the quartz mills. They are placed in the sluice below the battery in any place where there is not too great a current of water. They are about 3 in. deep and slightly curved against the current, and are made the width of the sluice. They are swung by wires passed through eyes in the plates and the sides of the sluice. The plate thus hangs across the current with its curve against it. It is half buried in the water. The force of the current keeps it swinging, so that the gold is forced against it. Immediately beneath the plate in the sluice a line of amalgam is formed which drops from the plate, and this and that on the plate is thus saved. These plates are hung a few feet apart and must be constantly examined.

The plates require "skinning" every three or six months, according to the quality of the material put through. The old manner of skinning with mallets or hammers is very objectionable, as with the utmost care the plates become dented and will no longer lie flat. It is better to remove them and heat them over bark burned to embers, or live coals taken from under the boiler grate. Two persons should pass each plate over the embers quickly and uniformly, in small sections of about 3 ft. square at a time. They are then placed on a mill table, or other flat surface with a wide margin, and scraped with chisels or stiff palette knives, heating and scraping rapidly until completed. In this manner a considerable quantity of amalgam not required on the plates is secured. After relaying, the plates should be rubbed with mercury to make them ready for use again. Outside plates are best

arranged in a system of steps, at an inclination of at least 3 in. in 12 ft. Generally the first table is made about 6 ft. long, and the rest about 30 in. long and 12 in. or 18 in. wide, the first plate being tapered to this width. Any number of these steps can be arranged according to the fall available, and preferably in zig-zag shelves, so made as to be interchangeable and easily slid out and in for cleaning and replacing by duplicates. Six shelves in use and one or two duplicates will generally be found to suffice.

Every battery is hung up two or three times a day to examine the screens, to see if any are broken, in order to replace them. The pieces of wood or any other material which are found in the mortar when the screens are taken out are removed, and the amalgamated plates, are examined and washed with water to which a little ammonia or potassium cyanide has been added for the purpose of removing the grease, which comes mostly from the oil of the machinery. When the ore has not been roasted a small quantity comes from the candles which the miners use; and in winter, when the exhaust steam is used to heat the water or the battery, it comes from that. Sometimes the copper of the battery plates, which are exposed to the air, is coated with a thin film of oxide, which must be removed. A new plate is not considered to be in a fit condition to be placed in the battery until it will remain bright for some hours, and this will not be until a certain quantity of gold amalgam has collected on its surface. This is so necessary that in coating a new plate the copper amalgam should be formed first, and it is then rubbed with gold amalgam. No amalgam is ever removed from the plates until a hard coating of gold amalgam has first been formed. It should always be allowed to accumulate on the aprons during the run unless the ore is very rich, as amalgam serves to catch amalgam. Both the grease and oxide coat the gold and plates and prevent the action of the mercury. Once, or sometimes twice a day, when the ore is very rich, at the end of each shift, generally in the morning, the sluice plates are cleaned. The plates of the splash box are cleaned when the battery plates are. They are first carefully rubbed with a piece of india-rubber belting, the edges of which are sharp, held between two pieces of wood, so that only about $\frac{1}{2}$ in. projects. The amalgam is removed with dull chisels or

scrapers, generally made of old files, great care being taken not to dent the plates. The whole of the plates are rubbed with a little fresh mercury, and again rubbed with the rubber, to make them perfectly bright, as the best result is obtained when the surface of the plate is perfectly clean. Four to five ounces a day are collected from here. Care must be taken in cleaning these plates to which the amalgam adheres, not only not to dent the plate but to leave a certain proportion of the gold amalgam on it.

From the battery sluice *b*, Fig. 53, the pulp runs into Hendy's

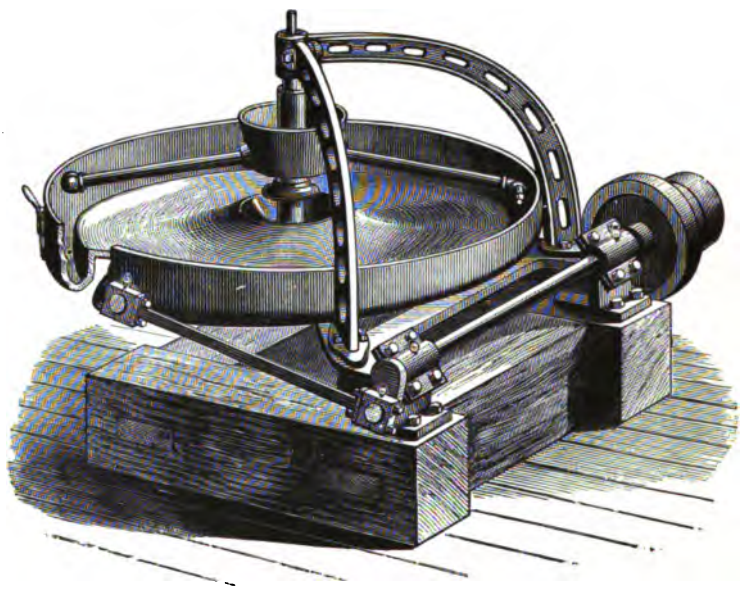


FIG. 54. HENDY'S CONCENTRATOR.

concentrators, Fig. 54. This is one of the oldest of those used in the Pacific States. It is now very generally abandoned, though it is still used by a few mills. It consists of a shallow pan, 5 ft. or 6 ft. in diameter, supported in the centre on a vertical shaft, made to oscillate back and forth by means of cranks on a shaft placed at one side and joined to the outside of the pan by connecting-rods. It is made of three different portions, an annular gutter, which is 2½ in. wide, commencing at one side at zero, and gradually deepening until at the discharge it is 2½ in. deep. The floor or bottom of the pan is curved. It is 21 in. wide

and rises in the centre nearly to the height of the sides of the pan; while inside there is a circular basin 14 in. in diameter. Through the centre of this basin there is a circular opening 2¼ in. in diameter, to receive the shaft, about which the pan oscillates. The pan turns on its vertical axis back and forth for a short distance at every revolution of the crank-shaft, so that a short, quick, vibrating motion horizontally is given to it while it is turning on the centre as a pivot. In its upper part the shaft passes through the ore hopper, and is supported at the top by two curved arms, which are bolted to a triangular frame. The pulp is delivered into the bowl above the centre and is carried by two arms, which are pieces of pipe, and delivered on the outer edge of the pan. It is necessary that this pan should be firmly fixed in the centre; otherwise it will quickly work loose from the oscillating motion. The pan is set up level, the levels being taken from the rim near the centre. The movement is communicated by means of two connecting-rods fastened to the outside of the pan, adjusted to crank-pins of peculiar shape, to prevent the action of the sand upon them. The crank moves the pan a short distance backward and forward at every revolution of the crankshaft. The ore is received from a spout into the revolving hopper, from which it is discharged into a T-shaped distributor, the main pipe being attached to a piece of pipe fitted to the shape of the outside of the pan 2 ft. long, with slots and holes on its top. To work it, it is filled about half full of the material to be concentrated, which will be about 3 in. On account of the motion of the machine, the heaviest parts will settle to the bottom and are gradually removed to the outside of the pan, where they collect in the trough provided for them. The tails collect on the bottom in the order of gravity, and are gradually carried up the incline of the plane, the lighter particles being discharged at the inside circular basin from the centre of the machine. The discharge gate at one side is opened to discharge the concentrates as they accumulate gradually. This must be regulated to correspond with the rapidity of the accumulation of the material at this point. If opened too wide, the sand from the layer above will be discharged, and if the opening is too small, the concentrates will

accumulate and flow over the centre of the bridge. Any amalgam or quicksilver collects in the outside depression. The concentrates are usually allowed to accumulate until the pan is about half full. The crank should make from 215 to 225 revolutions a minute, so that the sands are constantly shaken at that rate. Great care must be taken to see that there is no lost motion, since blows and jerks, resulting from loose joints, will make the machine wear very badly. When the machine is properly fed and in working order, the discharge of concentrators and tails will be continuous, but it must be regulated in every case until it has been ascertained how it will work with each particular material which is being treated. The concentrators are usually set in pairs, a single crankshaft being sufficient for both, and the driving shaft required being only 6 ft. in length. They weigh about 1000 lb. each, but smaller ones are sometimes built for prospectors or for carrying on the backs of mules, of a diameter of not more than 24 in., and work very well as prospecting tools. Once carefully levelled and set up, they require no further attention, and are not likely to get out of order, and then are easily repaired. It is very useful as a separating machine for the concentrates and amalgam. The amount of water which is required is not large, and each machine will easily treat 5 tons of crushed ore coming from the battery in every twenty-four hours. Eight tons can be treated in the same time, but it is not generally advisable, as from too rapid feeding there will be a certain amount of loss, and the concentrates will not be entirely free from sand. If the concentrates must be clean or are being prepared for sale, the concentration from four machines should be conducted through a fifth, so as to remove entirely all the sand which they contain.

The Duncan concentrator, like the Hendy, has a circular iron pan whose movements are intended to resemble that of a shovel when vanning by hand. These motions cause the heavier particles of the mineral to settle at the bottom round the outside, where it remains until discharged, while the gangue is held more or less in suspension, and is carried to the centre, from which it is discharged. The pan makes eight and a half revolutions per minute, the pulp makes about three revolutions in the pan, giving

time for the mineral to settle. The motion of the pan can be adapted to any ore by adjustments. This concentrator weighs about 1750 lb. It takes about half a horse-power to run it.

A centrifugal amalgamator was used at the Stickel's Mill. It consists of a copper pan 5 ft. in diameter, set on arms in an iron pan, which has a circular ring 12 in. wide round it. The inside of the copper pan is silver-plated and amalgamated. Above the pan, and fitting into it, is a stationary copper cover, which has a funnel for introducing the pulp to the middle of the centre ring, and three openings to let it out. The top is always stationary. The ring revolves at the rate of thirty-one revolutions per minute. It would probably work better if it went slower. The pulp from the centre is forced over the very large surface of amalgamated plate, and falls into the ring in the centre, from which it runs to the concentrating sluices. The results of a six weeks' trial with the machine was favourable.

B. The Frue Vanner.

The Providence Mine of Nevada City is an excellent example of ores the largest part of the gold of which is contained in the sulphurets, which amount to from 5 to 6 per cent. All the ore is hoisted without sorting, and is sent over a tramway directly into a rock-house in the top of the mill. There it is dumped upon an iron plate 4 ft. wide, and falls on to grizzlies 4 ft. wide, 12 ft. long, the bars of which are 4 in. deep and 1 in. thick. They are 2½ in. apart held in position by 1-in. rods, which are covered with ferrules between the bars, through which the fine dirt falls into an ore bin, while the coarse rock passes over them and is delivered to Blake's crushers, which have 10-in. jaws. In the coarse ore there are some pieces of nearly pure sulphurets, which are thrown to one side and treated separately; the rest of the coarse stuff is fed into the rock-breakers and falls from these directly into the same bin that holds the fine dirt. From this bin all the ore passes down chutes to automatic feeders which deliver it to the stamp batteries. The battery screens are No. 5 punched. From the battery aprons the pulp runs over silver-plated copper plates to Frue concentrators.

The mill has forty stamps of 750 lb., which have a drop of 6 in. to 8 in. It makes from ninety-three to ninety-six drops a minute. The shoes weigh 125 lb. and the dies 95 lb. They are made of chilled cast iron. The shoes last from thirty to thirty-five days and the dies from forty to sixty-five days. Steel shoes last ninety days. Each battery is fed by a Hendy's Challenge feeder. The stamps crush 2 tons per stamp. The pulp is quite coarse, because of a low discharge and a high speed; but the free gold is effectually released, and is caught in the battery and on the plates, while that contained in the sulphurets is recovered from the concentrates.

The amalgamating plates are 4 ft. long and the width of the mortar, inclined $1\frac{3}{4}$ in. in 1 ft. This rather steep pitch is employed in order to keep the plates clean and yet avoid the use of much water which would flood the concentrators. Water is used in the batteries in quantities only barely sufficient to keep the plates clean at this inclination; if there is an occasional banking up of sulphurets on the plates the latter are readily cleansed with a hose. On the washing field of each Frue vanner just enough clear water is used to prevent the gangue from being carried with the concentrates over the head of the machine. Great care is taken to have the concentrators perfectly level and their lateral motion properly adjusted, to prevent the sand from banking on either side of the travelling belt. The tailings from the concentrators, being practically barren, are run into the river; the concentrates are remarkably clean, containing as a maximum only 10 per cent. of sand. They are spread on the floor of the sampling-room to dry. When the coarse pieces of sulphurets, picked from the ore in the rock-house, have accumulated in sufficient quantity, they are put through a battery separately and from the battery plates the rich sulphuret pulp is run directly into settling tanks, from which it is afterward shovelled, and mixed with the concentrates on the floor of the drying-room. The dried sulphurets contain $120 in gold and $12 in silver per ton, and are taken to the chlorination works, where they are roasted, chlorinated, and leached.

They use 80 in. of water in twenty-four hours at a pressure of 380 ft. The mill is run by a 6-ft. Pelton hurdy-gurdy wheel

driven by 104 in. of water under a head of 390 ft.; 5 in. of water are used in the batteries and concentrators and chlorination works, and one is lost, making 104 in. used for the power to drive the mill. The water costs 16 cents per inch, or $16.64 for twenty-four hours. It would require 12 cords of wood to furnish the same amount of power. They get 20 per cent. of gold in the batteries and 80 per cent. on the plates. The value of the sulphurets is $130 and the cost of treating them $6. The large pieces of pyrites found in the ore are put on one side until there is a sufficient supply, when they are crushed separately in the battery and mixed with the concentrates for treatment. The concentrated pyrites are roasted in two elliptical furnaces whose capacity is 4.5 tons each in twenty-four hours. They are then chlorinated and leached for gold. The residues are leached with sodium of hyposulphite for silver. The cost of the treatment for an output of 9 tons in twenty-four hours is $3.55 per ton.

Eight men run the mill day and night and keep it in repair; two men by day, and one at night, spall the rock and feed the crushers; three men by day, and two at night, take care of the batteries, tend the concentrators, and deliver the concentrates into the drying-room. The daily running expense then would approximate the following:*

			$
3 men at rock-breakers at	$2.25	...	6.75
3 ,, ,, batteries ,,	2.25	...	6.75
1 man ,, concentrators ,,	2.50	...	2.50
1 foreman ,,	3.50	...	3.50
110 inches water ,,	.16	...	17.60
Wear on shoes	...	...	6.11
,, dies	...	...	3.42
,, screens	...	...	3.87
,, rock-breakers, ,, concentrators, ,, copper plates	...	...	5.00
Illuminating	...	...	1.00
Lubricating	...	...	1.00
Total	...	...	$57.50

The average quantity of ore milled per month is 1850 tons, or 62 tons per day, which brings the cost for milling and concentrating to about 93 cents per ton.

The Sierra Buttes Mine, in Sierra County, California, is an excellent example of a low-grade mine, which has been worked

* School of Mines Quarterly, vol. v., page 370.

since 1851, and under good management has yielded a profit. The crop of the mine is situated at an altitude of 6400 ft., the winters are cold and the snowfall heavy, and transportation difficult and expensive. Automatic machinery is used everywhere throughout the mine and the mill. The mill is capable of crushing 4500 tons per month, and cost, including the chlorination works, $100,000. The average yield of the ore in 1882 was in free gold $6.47, with 59 cents in the tails as pyrites, or a total of $7.06. The cost of mining was $4.10, and of milling 60 cents, or $4.70. The tails were worked up in arrastras, of which there were formerly thirty running. They have since been replaced by Frue vanners. In 1883 the cost of mining was $4.98, and of milling 58 cents, or $5.56. The yield of the ore was $7.48 in fine gold, and 70 cents in the tails, or a total of $8.18. In 1884, during the first six months the yield of the ore in free gold was $7.35, and 17,841 tons were treated. The cost of mining and milling together was $4.93. The yield of the ore might have been made very much higher by hand-picking, and this would have been good policy if the supply of ore had been limited, but it has been the principle of the company to reduce all ore that would pay enough to treat, so that the yield per ton was consequently lowered; but the policy has proved to be an excellent one. In 1888 the ore contained ½ per cent. of sulphurets, which, when concentrated, are worth $97 in gold and $3 in silver to the ton. These are treated by Plattner's process at the mill, which has taken the place of the arrastras formerly used. The ore is treated by battery amalgamation and concentration. The mill has two stamp mills, one the Hitchcock, with fifty stamps, which, on account of its elevated position, is not easily kept supplied with ore; and another the Yuba, with sixty stamps, which was the only one running in 1888. The mill was formerly run by steam. It is now run by water power, the supply being drawn from several lakes. It is brought around the mountain in flumes 7½ miles long, which supply 400 in. of water under a pressure of 550 ft., which can be increased if necessary to 1500 ft. The mill uses 125 in. of water.

The ore is contained in a quartz vein, varying in width, and containing in 1887 $4.17¼ in gold with a little silver and a trace

of copper and one-half per cent. of sulphurets. The ore comes from the mine, and is dumped automatically over grizzlies into a bin from which the mill cars take it. The bin capacity of the mill is 1000 tons. It has three Blake's crushers, one in the centre and one at the end of the battery line. Only one of these runs at a time. When its bin supply is worked up, another crusher is run. Water power is used; 320 miners' inches of water are used at a pressure varying from 60 ft. to 620 ft., which will average 550 ft. The mill has upon one side a 6-ft. Pelton, and at the other side a 6-ft. Knight's wheel. The Knight's nozzle acts upon several buckets at a time, and has a slot $\frac{5}{8}$ in. by $1\frac{1}{2}$ in., and wears rapidly. The Pelton has a $\frac{15}{16}$ in. round nozzle. The Knight's wheel runs twenty stamps and one breaker, and the Pelton forty stamps and two breakers. Either wheel could run the whole mill if necessary. The water, brought in a ditch seven miles long, the most of which is flumes, is used at a pressure of 550 ft. The mill has sixty stamps of 850 lb. The stem weighs 365 lb., the boss 180 lb., the shoes 160 lb., and the tappets 145 lb. They drop $7\frac{1}{2}$ in. to 8 in. and make eighty-seven drops. Each stamp crushes $2\frac{1}{3}$ tons in twenty-four hours. The shoes are of steel and last 100 days. The dies are cast iron. They weigh 100 lb., and last forty-five days. The feeders are the Challenge. Each ore bin holds 140 tons. The mortars are lined at the end and back with liners $\frac{3}{4}$ in. thick, which last nine months. On the back plate is an amalgamated plate 4 in. wide and the length of the battery. The splash plate inside is also plated. The screens are No. 7 punched slot on the end batteries, and No. 5 in the centre. They are brushed over occasionally from the outside with a whisk broom to keep them from clogging. The apron is only 1 ft. wide, and is really only the mortar lip. It is closed by a board with a curtain as is usual in most of the mills. It connects with a distributing box, and runs from there to the sluices. In front of the batteries are sluices set so that the adjoining batteries run their pulp together. These sluices are 16 ft. 6 in. long and 14 in. wide. The plates on them are dressed four or five times in twenty-four hours. There are three plates lapped in the length of the sluices. In some cases one of the lower plates has been cut in two so that the lower plate is lapped twice. Silver-plated plates cannot be

used in this mill. The silver is worn off so rapidly that the constant silver plating would cost too much. The cause of this wear is attributed to the action of the small quantity of galena and blende; 80 per cent. of what is saved is caught in the battery, and 20 per cent. on the plates. The total length of sluice plates in the mill is 240 ft. The overflow from the plates goes on to twenty-four Frues, which catch three-quarters of the sulphurets, which yield from $70 to $500 to the ton. The Frues use 4 in. of water. There are amalgamated plates on the feeders. The Frues are run with a 5-ft. Pelton wheel with a 1½-in. bucket and a ½-in. nozzle. Only ten men are employed in the mill. Underneath the Frues is a system of catch-boxes. The catch-box is divided into three parts, the two ends of which run the whole width of the box and the middle ones are divided into three parts. What overflows into the three divisions of the middle compartment is worth $500 to the ton. Very little is accumulated in the six weeks. The fine material concentrated on the Frues is worth from $80 to $125 per ton.

The mill is lighted by electricity. The dynamos are run by a 6-in. Knight's wheel made of brass. The water which supplies the mill comes in a 24-in. pipe reduced to 14 in., made of No. 8 iron. The pipes are in pieces 16 ft. long, and are muff jointed. All the muffs are leaded. All the sand which accumulates in the mortar, and that which is found in the end boxes of the sluices, which is called "riffle stuff," and the amalgam is treated in an iron barrel 5 ft. long by 2 ft. 6 in. in diameter, which is made to revolve twenty times a minute for twenty-four hours with mercury added. It is then turned out on a batea 42 in. in diameter, in which three iron balls 4 in. in diameter revolve. The amalgam is treated separately, but in the same way. The cost of milling in 1887 was $0.3925 per ton.

The Zeile Mill of Amador County, California, is an excellent example of a mine with a very poor ore which has been successfully worked by good management. The ore is talcose slate and quartz, yielding about $2 in gold and $5 to $6 per ton in sulphurets. It will average from $3 to $6 total gold. The ore from the mine is dumped to the crusher and goes through it to the ore bin without any labour at the crusher. The mill is

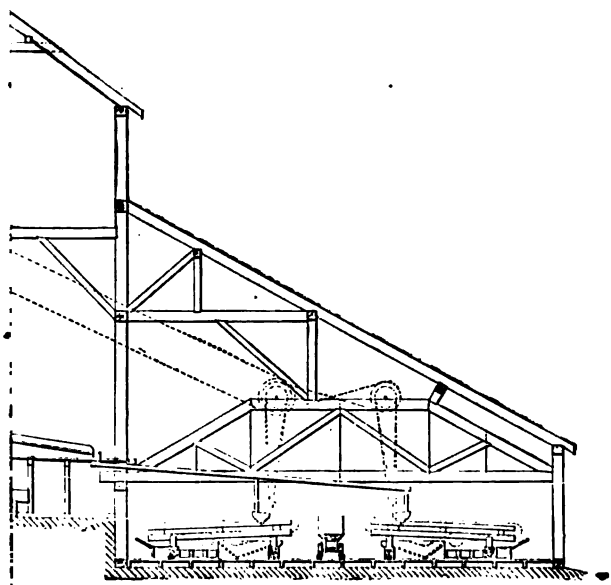
arranged both for steam and for water. When water is plenty a 6-ft. Pelton wheel is run, using 180 in. of water with 150 ft. pressure. The power is transmitted by Manilla ropes. The batteries are fed by the Challenge feeder. The mill has forty stamps of 750 lb., which make eighty-eight drops of 7½ in. The order of the drop is 1, 5, 2, 4, 3; 15 in. of water are used in the battery. The stamps crush 3½ tons in twenty-four hours through a No. 4 straight slot. The shoes and dies are of chrome steel. The aprons are 56 in. by 36 in. with an inclination of 2½ in. The sluices are 14 in. wide and 12 in. long with an inclination of 1¼ in. to the foot, and are covered with a single plate. This is followed by a sluice of the same width 4 ft. 6 in. long with two riffles, which are respectively 8 in. and 16 in. from the top. Four men and one foreman, who have no other duties, run the mill. Thirty-three per cent. only of the gold contents of the ore is saved in the batteries and the plates. From here the pulp is distributed on to the Frues, which use 5 in. of water, so that 20 in. are used in the mill. The gold caught in the mill is worth $17.75 per ounce. The concentrates are worth about $100 per ton. From the Frues the tails run on to four sets of four sluices 50 ft. long or 48 ft. of available length below the head boxes which supply the clear water. These upper sluices are 12 in. wide. At the end of 50 ft. the sets of four sluices become four sets of two with their own head box for clear water. At the end of the first 50 ft. there is an undercurrent for the reception of the concentrates, and the same at the end of the second set. The sluices are laid with wire cloth twenty-four to the inch and bagging, which acts on the same principle as the blanket. The amount which had been collected in these tail sluices up to August, 1888, had not paid days' wages, but there was a fair prospect of its doing so. The round buddle which was formerly used both here and at the Stickels Mill has been abandoned in both places.

At the Stickels Mill the tails from the battery sluice discharge on to two blanket slucies 12 ft. long and 16 in. wide, and from thence to a set of six sluices 80 ft. long and 12 in. wide. These sluices are riffled every 16 in. Four of these are used for catching the pulp, and two for concentrating it. The pulp from the battery runs into a distributor, over which there is a head box

for clear water. In the four sluices a riffle ¼ in. high is placed every half-hour. From the end of these sluices they discharge into another set of two sluices 80 ft. long and 12 in. wide with riffles every 16 ft. These riffles are put in every six hours. The four sluices above are cleaned into the two concentrating sluices every day. The material collected in the lower is concentrated in the upper sluices. That from the blankets is put into an iron cylinder 48 in. by 24 in. with small pieces of iron and mercury, and worked for twenty-four hours at the rate of forty revolutions per minute. The residue is worked in a pan.

At the Spring Valley Mill in Amador County, and in most of the other mills, the dressing of the sluice plates is done once a day. For this purpose the pulp is turned off from the battery, and a stream of clear water allowed to run on it long enough to wash off all the pulp. The ore is contained in a slate, which on crushing makes the water entirely black. The plates are then washed with a sponge, beginning with circles from the top towards the bottom and then returning from the bottom to the top, to wash off any material which may have remained. It is then washed with an ordinary whisk broom, brushing it from the bottom towards the top, a little water being dipped out from the pan in which the utensils are kept, in order to make the washing complete. The plates are then scraped with a rubber scraper in a handle, scraping from the end up and from the top down. The amalgam is collected with a scoop and put into the utensil-pan and covered with water. Mercury is then sprinkled from a bottle with a cloth over the mouth for 2 ft. or 3 ft. down the plate. It is then rubbed down with a sponge, turning it constantly around from the top half way down. It is then rubbed over with a whisk broom, working from the top towards the bottom, and the pulp from the battery is then turned on again.

The Treadwell Mill on Douglas Island, Alaska, Fig. 55, treats a quartz containing fine gold with sulphurets, which is quarried in deep open cut near the mill. The ore runs from $5 to $6 per ton. The ore from the mine is dumped on to grizzlies; what passes them goes to the ore bin, which holds 2000 tons, the rest goes to the crushers and then to the ore bin. The mill has 240 stamps arranged 120 back to



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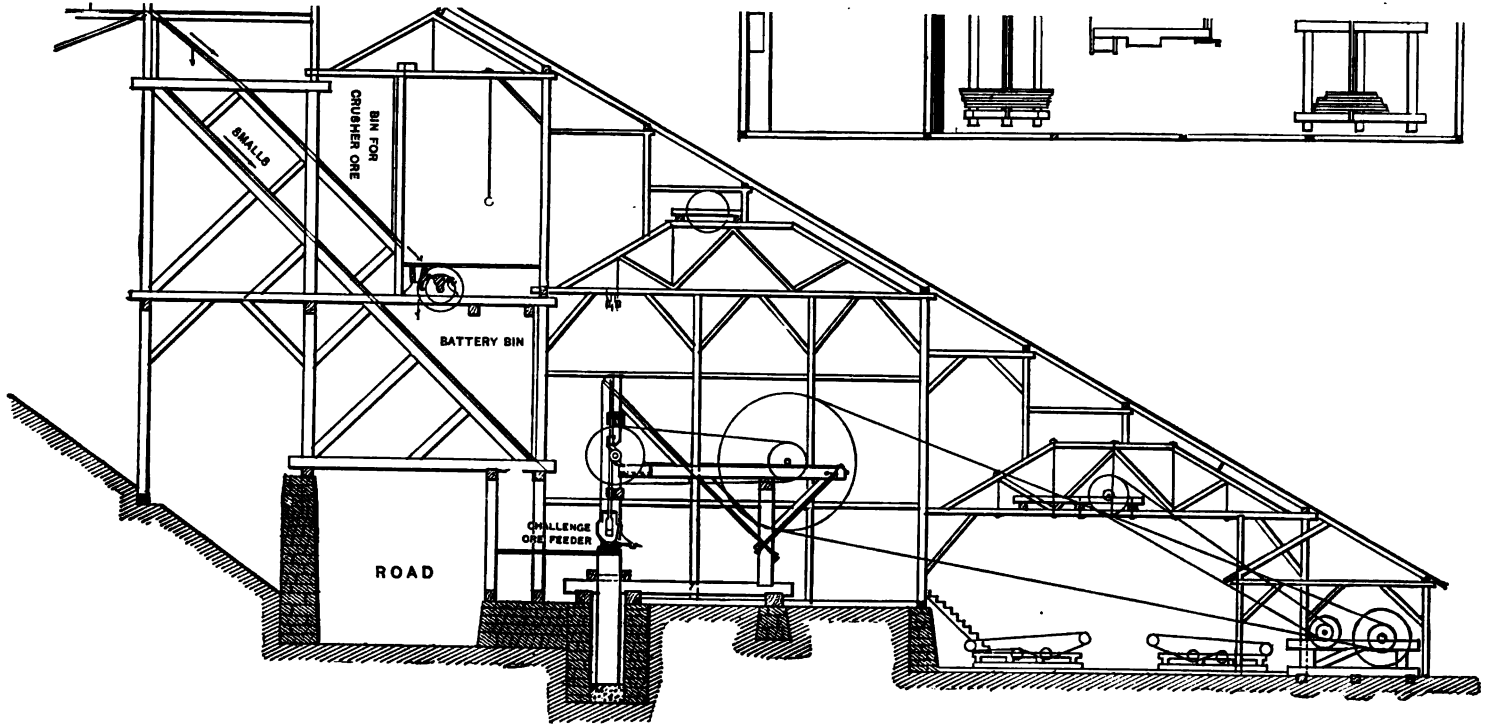


FIG. 56. NORTH STAR GOLD MILL.

back; for these there are six crushers capable of crushing 100 tons a day. The crushers are on the Blake system, and have the shoes and dies bedded in lead. The batteries are arranged in sets of four. The stamps weigh 900 lb. The stems 350 lb., the tappet 140 lb., the boss 250 lb., and the shoe 140 lb. They make ninety to ninety-two drops of 8 in. Each stem has its own guide, so that it can be taken out separately. They crush 2½ tons in twenty-four hours. The shoes and dies are made of chrome steel, and last on an average 250 days. Amalgamation is done in the battery. The screens are No. 8 slot. They discharge on to amalgamated plates of three on each sluice. The sluices are 12 ft. long and 2 ft. wide. From these the sluices run into a short sluice with a mercury trap, and then discharge on to the Frues, which make 200 side strokes per minute. There are two for each battery. The battery plates are dressed every twenty-four hours by causing clear water to run over them to clear off the sand, and then rubbing them with a hard brush and a rubber, and sprinkling mercury over them. They are cleaned once a week. A general clean-up is made once a month. There are no amalgamated plates on the Frues. The concentrates from the Frues run from $80 to $100, and are treated by chlorination in the company's works. Twenty-eight men were required for the first 120 stamps; only twelve more, or thirty in all, are required for the 240-stamp mill. The cost of mining and milling is $1.25. The mill is run by one 6-ft. Knight and one 6-ft. Pelton wheels.

The North Star Gold Mill in California, Fig. 56, which was built in 1887, is another of the type of mills in which battery amalgamation and concentration is used and also water or steam as a motive power. The water is brought to the mill in 24-in. wrought-iron pipes 10,000 ft. long, with a pressure of 270 ft. At the mine it branches into two, one 16½-in. pipe going to the mill, and a 14½-in. pipe to the hoisting works. The pipe line is coated with asphalt and tar. The ore from the mine is dumped over grizzlies into the ore bins. The coarse ore is fed into two Blake's crushers, and the two collect in bins which supply the stamp batteries. The motor for the mill is one 6-ft. Pelton wheel for the battery line shaft, the power being communicated by four

2-in. hemp ropes. These are provided with hydraulic governors to secure uniformity and efficiency in the mill. Counters are attached to the shaft and a hydraulic pressure gauge shows the variations of pressure in the pipes. The rock-breakers are run by one 4-ft. Pelton wheel connected by a 5-in. and ⅝-in. wire rope. The concentrators are run by a 3-ft. wheel transmitted by 1½-in. hemp rope. These have also governors, counters, and gauges. These motors are equipped with deflector nozzles and a Moore's differential hydraulic gate. The stamps are arranged in six batteries of five stamps with single discharge motors, and automatically fed by six Challenge ore-feeders.

The mill runs on a quartz containing $15 to $20. The tribute system is extensively used in the mine. The sulphurets, which run from $50 to $75, are from 3½ to 5 per cent. of the ore They are concentrated on the Triumph concentrator. They hoist 80 tons a day. The crushing bin contains 500 tons, the feeder bin 500 tons, or a total of 1000 tons storage capacity. There are two rock-breakers, 9 in. by 15 in., Blake's improved. Five bins of ore are used for the company's work and one bin for the tributers. This makes six bins for the whole mill. There are thirty stamps, in six batteries of five each, one battery of which is used for tributers alone. The stamps weigh 850 lb., drop 6¾ in. to 7 in., and make eighty drops a minute, and crush 2 tons in twenty-four hours. They weigh exactly as in the Idaho. The shoes, dies, cams, and tappets are made of cast steel. The dies last eighty-five days. The mortars have back and side liners. Amalgamation is done in the battery and on the plate in front. The screens are brass wire cloth, thirty to the mesh. The quantity of water used in the batteries is seven miners' inches for the thirty stamps. The apron plates are 57 in. by 54 in. There is then a drop of 1 in., then a space of 6 in., with a wooden piece with 20¾-in. holes, from which the pulp falls. The plates on the battery sluices are 4 ft. long and 4 ft. 6 in. wide, in 2 ft. going down to 22 in. They discharge through 20¼-in. holes in a wooden plate. The battery plates are 12 ft. 6 in. long and 22-in. wide, made of three plates which lap. The pulp then falls on to shaking tables. which shake backwards and forwards sixty times a minute These are 2 ft. wide at the top, and in 3 ft. widen to 46 in.

They are 12 ft. long. They have four copper plates of equal size lapped, and 2 ft. from the end the sluice is bare. All of these sluices are of wrought iron, both the battery and the shaking. The sluice ends in a sand-box, and goes from there to the Triumph concentrators, of which there are two for each battery. A space has been left in the mill for ten more stamps and their appendages. The material taken from the sand-box is panned by hand. The plates are dressed every day. The clean-up is made twice a month. The sand from the amalgam and that from the sluices is cleaned in barrels with old pieces of iron. The concentrators have a feed-box with a set of wheels to move the sand, which revolves seventy-eight turns a minute. There are no amalgamated plates on the concentrators. They make 230 end strokes a minute. The belt travels at different velocities, according as it is loaded. The water from the fresh-water box discharges 12 in. from the head. The ore discharge has twelve openings $1\frac{1}{4}$ in. by $\frac{1}{2}$ in. The Triumph concentrator is very similar to the Frue, but has an iron frame and no side shake. The motion to the belt is given by a leather friction wheel. The sulphurets are collected in a room made for the purpose, and are piled up until they are about 30 in. thick. They are dried here until the moisture is about 2 per cent. in summer. Their value is determined by assay. The sample is taken with a probe, which is $\frac{3}{4}$-in. tube cut in half in the direction of its length, with a handle across the end. It is 4 ft. long. The samples are taken every 6 in. across, and in the length of the pile. They are all sold to the chlorination works in the vicinity. They get 92 per cent. of their gold contents for them, and in addition, a charge of $12 to $15 is made for the treatment. No sulphurets are allowed to the tributers. Eighty per cent. is saved in the batteries, and twenty on the plates. The bullion is 850 fine in gold and 150 in silver. Six men are employed in the mill. The total amount of water used in the mine and the mill is 312 miners' inches. The head of water is 270 ft. It has previously been used in the Empire Mill. The water is from the South Yuba Water Company, and goes to the Empire under a head of 440 ft. It is delivered from the mill at a pressure of 270 ft. and then goes to be used in agriculture. The milling costs 70 cents. The

loss in mercury is 1 oz. per ton. For cleaning up, the mill has a cleaning barrel, batea, and clean-up pan. The storage-room for drying and sampling the sulphurets, and the retort-room, are placed near together. The clean-up room contains the cast-iron cleaning tanks, which are covered with marble slabs, the washingstands and the clean-up pan for the mercury. The mill water is taken from a tank, being supplied by the tail-race in the hoisting works by the centrifugal pump and heated to the temperature of about 100 by a jet of steam from the boiler. From the tank a 4-in. pipe leads the water to the mill, where it is distributed by smaller pipes to the batteries, concentrators, and clean-up machines.

The Empire Mill has a quartzose ore averaging in gold about $14 with $2.25 in the sulphurets. The concentrates run from $125 to $300 to the ton. The percentage of sulphurets is 3½. The ore bin below the rock-breakers holds 700 tons. There are two rock-breakers, 9 in. by 15 in., which discharge into Challenge feeders. The mill has forty stamps of 850 lb. The shoes and dies are of chrome steel and last four to five months. The shoes weigh 155 lb. The stamps make ninety-four strokes of 7 in. per minute. They use No. 30 wire screens and crush 2¼ tons each. They amalgamate in the battery on a plate 4 ft. by 4 in. The battery apron is 5 ft. by 5 ft. 2 in., reduced to 22 in. The sluice is 12 ft. long by 12 in. wide. The whole length of the sluice is 20 ft. 6 in. All of them are made of wrought iron, they are all copper-plated, and the plates lap. The inclination of the plates is 8½ deg., 7 deg., and 6 deg. From the battery plate to the apron the fall is 4 in.; from there, 2 in. The sluices end in a Hungarian riffle, which is 24 in. wide and 13 in. deep, in which a little mercury is collected. The plates are dressed every twenty-four hours. Eighty per cent. of the gold recovered is caught in the battery, and 20 per cent. on the plates. The cost of mining is $3.00, and of milling $0.75.

The pipes to the Triumph concentrator are 2½ in. in diameter. There are two concentrators to the battery. The speed of the belt is 3 ft. per minute; the number of strokes, 235. In the concentrators the experiment is being made of washing the concentrates off the belt by a stream of water directed at right angles to the belt

half way down the side to the end. To do this a piece of rubber cloth is set up against the belt so as to press against it. Water is directed against the belt and a little above it, to wash the sulphurets off. The holes of the iron water-supply pipe were constantly becoming closed, so that no water came in on the belt at that point, and consequently the sulphurets were found in two places, under the machine, from which it is difficult to remove them, and in the wash-tank in front of it. The washing would probably have been more effective if the stream had been directed down instead of at right angles to the pipe. The pipe was too far away from the belt, so that the discharge was constantly being interrupted. Seven men are employed in the mill. It is run by a 6-ft. Pelton wheel, with a $\frac{15}{16}$-in. nozzle; the concentrators are run by a wheel 2 ft. in diameter. The amount of water used by them is 18 in., under a pressure of 475 ft.; 80 in. of water is used in the mill; and 150 in. in the mine, under a pressure of 440.

The clean-up is made every two weeks. The amalgam is cleaned in a Knox pan. The battery sand is panned and put through the battery again. The amalgam is cleaned in an arrastra with slack lime, four to five handfuls of lime being used for every 2000 oz. of amalgam. The arrastra used for cleaning it is 2 ft. 6 in. in diameter and 18 in. deep, with two drags of 25 lb. The tails run over sluices 2 ft. wide and 65 ft. long, with riffles every 4 ft. There are three of these sluices side by side. The riffles are changed every twenty-four hours. They are cleaned up every day and ground in a pan. It hardly pays expenses. A sample of the tails is taken by clockwork, which moves a gate so that every hour a certain portion of the tails go into a box for a certain number of minutes.

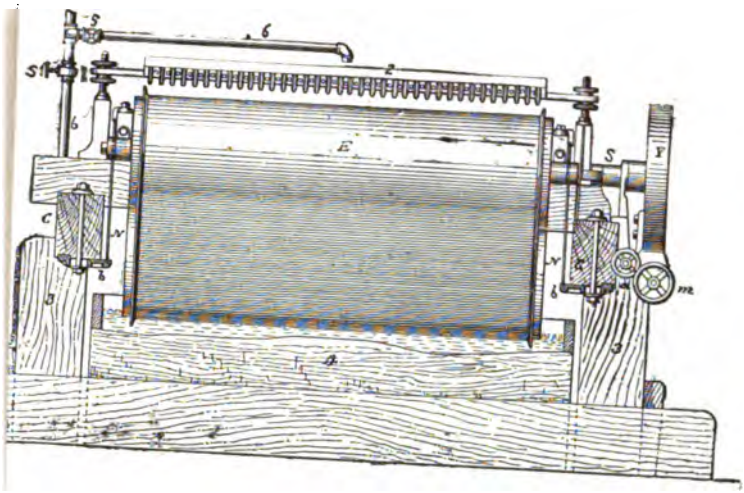
The Frue vanner,* Fig. 57, has of late years been introduced in a very large number of the gold mills in the United States. It has found many imitators of the principle on which it is constructed, some of which have become successful machines. The idea of using a revolving belt, either with or without riffles or buckets, as a carrier for various materials, is very old, but the

* For this information I am indebted to Mr. Walter McDermott, of New York.

application of a lateral shaking motion to a belt, a motion which closely resembles that given to a shovel when vanning by hand, is new, and is the principle upon which this machine is constructed. A revolving belt of canvas with a lateral or end blow has been used abroad and also in the United States, and though fairly good results were obtained from it, the constant wearing out of the belts caused them to be abandoned. The side shake is as much an improvement over the blow as the perfect rubber belt with permanent high flanges on both sides is superior to the rough, expensive, and short-lived canvas belt. The belt, which is the principal part of the machine, is made of rubber, inclined about ¼ in. to the foot, but the inclination can be changed according to the nature of the ore, so as to make it greater or less. It has also a lateral motion of 1 in., which is given to it by a crankshaft running from 180 to 200 revolutions per minute. The belt, counting the double length and the half circumference of the rollers, is 27½ ft. long. It is supported by rollers both at the ends and below. Its usual upper surface is 4 ft. wide and 12 ft. long, with rubber flanges on both sides. It is inclined towards the lower end, and travels up an incline, around two drums; the lower surface of the belt dips into a water tank, where whatever mineral is collected upon its surface is discharged by passing through the water. The belts have lately been made 6 ft. wide for the Silver King Mine in Arizona, where they have answered very well.

The ore is fed from the feeder, Fig. 58, by a stream of water, about 3 ft. from head of belt, and flows slowly down the incline subject to the steady shaking motion, which deposits the mineral on the belt. At the head of the belt is a row of water jets. The slow upward travel of belt brings up the deposited mineral, and the water jets wash back the lighter sand, letting only the heavy mineral pass and be deposited in the water tank below.

A A, Fig. 57, are the main rollers that carry the belt and form the ends of the table. Each roller is 50 in. long and 13 in. in diameter, made of galvanised sheet iron. The bolts which fasten the boxes of the rollers A A to the ends of the frame F also fasten to F the upper supports which rest on uprights N. The lower rollers B and C are of the same diameter, and are made in the same way as A A. The roller part of C is shorter than that



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of A A and B, and also has rounded edges, the upper surface of the belt with its flanges passing over it. The belt E passes through water underneath B, depositing its concentrations in the box 4, and then, passing out of the water, the belt E passes over C, the tightener roller; the rollers B and C are hung to the shaking frame F by hangers P P, which swing on the bolts fastening them to F. By means of the hand-screws, B and C can be adjusted on either side, thus tightening and also controlling the belt. The boxes holding the axes of the rollers A A in place have slots and adjusting screws, so that by moving them out or in, the rollers A A can be made to create a very strong influence on the belt E; and as it sometimes travels too much towards one side, this tendency can be stopped most quickly by lengthening or shortening on one end or the other of the rollers A A. The swinging of B or C also controls the belt. *c c* are bolts and washers to take up end play of rollers A A; these bolts pass through holes in the gudgeons of A A.

D D, &c., are the small galvanised iron rollers underneath the belt, and their support causes the belt E to form the surface of the evenly inclined plane table. This moving and shaking table has a frame F, of ash, bolted together with the rollers A A at its extremities. This frame is braced by five cross-pieces. The bolts holding the frame together pass through the sides close to the cross-pieces; the cross-pieces are parallel with the end rolls A A and the under rolls D D, &c., and their position can be understood by the three flat spring connections R O, &c., which are bolted to three of them, one to each, underneath the frame.

G G is the stationary frame which supports the whole machine. It is bound together by three cross-timbers, which are extended on one side to support the crankshaft H. The grade or inclination of the table is given by elevating or depressing the lower end of G G. This is accomplished by means of wedges; for this frame rests on uprights 3 3, fastened to two sills, which form the foundation of the machines in the mill.

F is supported on G G by uprights N, &c.; four on each side. These uprights are of flat wrought iron, with cast-iron bearings above and below, each middle bearing on F has one bolt hole, and there are two of them on each side. The end ones have two bolt

holes, and there are four of them, two on each side. These bolts pass through the frame F, and also hold to the frame the bearings of A A, which work in a slot. The bearings of the upper or head roller, are higher than those of the foot roller; *i.e.*, the upper roller is slightly higher than the regular plane of the table, and the first small roller D should be raised correspondingly.

The shape of the lower or bottom bearings of the uprights N, &c., can be understood by examining *b*, as shown in the end elevation and partly in the elevation. This lower bearing *b* extends across G, underneath, and is supported by a bolt passing through G. A lug on the upper side and on the outside end of *b* rests on G; and *b* hangs on the head of the bolt, and is kept stationary by the weight of N and its load. By striking with a hammer the face of *b* shown in the elevation, it is moved, changing the position of the lower bearing, and thus making N more or less vertical. By thus moving the lower supports of N, &c., the load and sand corners on the belt are regulated.

The cross-timbers binding together G G and resting on them are extended on one side, and on these extensions rests with its connections the main or crankshaft H. This crankshaft has its bearings X X X, on which are brass cups for the lubricating compound; the cranks are ½ in. out of centre, thus giving 1 in. throw.

I is the driving pulley that forms with its belt the entire connection with the power.

J is a cone pulley on the crankshaft H. By shifting the small leather belt connecting J and W the up-hill travel of the main belt E is increased or diminished at will. The small belt connects to J the flanged pulley W, which is on the small shaft K, and by means of the handwheel can be shifted on K and held in place. The bearings of K are fastened to Y, which is a cast-iron shell protecting the worm Z and the worm gear L; Y turns on a bearing bolted to the outside of G, and thus becomes a fulcrum for W and K. The object gained by this is that the weight of W and K (from Y) hangs on the small leather belt, preventing slipping or wear, at the same time making it positive.

a is a screw used to relieve the small belt from the weight of K and W, taking all the strain off the small belt, and thus stopping the up-hill travel instantly when desired.

m is a hand-screw by means of which the pulley can be moved, adjusting the small belt on the cone J, thus regulating the up-hill travel.

K is the wormshaft, and terminates in a worm Z, which connects with a worm gear L, which travels in a bearing bolted to the outside of G. Z and L are protected from dirt by the shell of cast iron Y enveloping both.

The short shaft which L revolves terminates in an arm S which drives a flat steel spring M, which is a section of a circle, connected with the gudgeon of A.

R, &c., are three flat steel spring connections bolted underneath the cross-pieces of F, and attached to the cranks of the shaft H by brass boxes O, &c. These springs give the quick lateral motion—about 200 a minute.

Q Q are two flywheels.

V V are two rods passing from the middle cross-timber to the lugs for same at the foot of F. The cast-iron washers on the bolts of the cross-timbers have lugs cast on them. V V pass through these lugs, and at each end are nuts on each side of the lugs. Thus V V prevent the movable frame N from sliding either up or down, and by them F is squared.

The belt E passes through water underneath B, depositing its concentrations in the box 4; and then, passing out of the water, it passes over C, the tightener roller.

2, above over the belt, is the clear water distributor, and is a wooden trough which is supplied with water by a pipe. The water discharges on to the belt in drops by grooves 3 in. apart. Another form generally used for 2 is an iron trough with brass spouts 1½ in. apart, so that when it is necessary by blocking every other hole, water jets can be made 3 in. apart.

1 is the ore-spreader, which moves with F, and delivers the ore and water evenly on the belt. The present form of the ore-spreader is shown in Fig. 58, and was devised so that for gold ores a silver-plated copper plate could be placed over the entire surface of the spreader, the top board B, with blocks attached, being put on the plate. C is a copper well that fits into and shakes with the ore-spreader. This is used in concentrating gold ores, for saving amalgam and quicksilver escaping from the silvered plates above,

and can be taken out and emptied at any time. Into this well all the pulp from the battery falls. Its ends are lower than the wooden blocks of the spreader, so that the pulp passes over the ends of the well and is evenly distributed.

The ore with water is delivered on the belt by means of the spreader 1, Fig. 57, and A, Fig. 58, which shakes with the table, and distributes the pulp uniformly across the belt. A small amount of clean water is distributed by 2, which is a wooden trough in which is a perforated pipe. A depth of $\frac{3}{8}$ in. to $\frac{1}{2}$ in. of sand and water is constantly kept on the table. The main shaft H should be given the proper speed for each kind of ore, and once established, it should be kept uniform; this speed will be between 180 and 200 revolutions of the crankshaft per minute, with 1 in. throw.

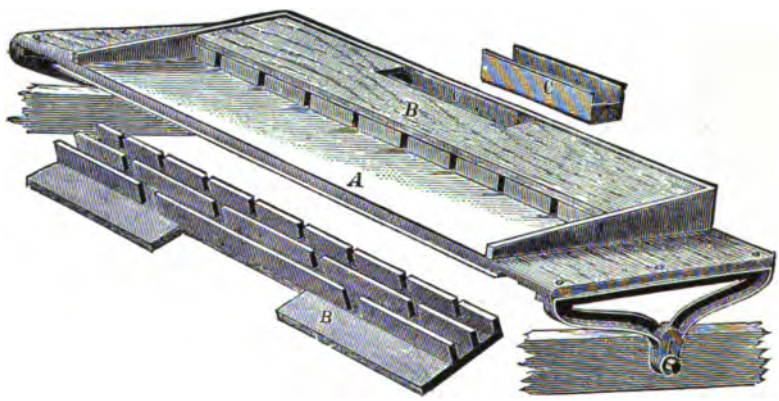


FIG. 58. FRUE ORE-SPREADER.

The up-hill travel or progressive motion varies from 2 ft. to 6 ft. a minute, according to the ore; and the grade or inclination of the table is from 3 in. to 6 in. in 12 ft., varying with the ore. The inclination can be changed at will, by wedges at the foot of the machine. The motion, the water used, the grade, and the up-hill travel should be regulated for every ore individually, but once established no further trouble is experienced in the manipulation.

5 5 are the cocks to regulate the water from the pipes 6 6.

3 3 are upright posts, which are firmly fastened into two sills. These posts are cut down on the inside to make square shoulders, on which G G rests.

The overflow from 4 contains finely-divided sulphurets in suspension; to settle them, the water passes through boxes 7 7 7.

8 is a section of the launder to carry off the tailings.

9 is a box into which the concentrations fall when scraped out of 4.

10 and 11 show the arrangement of the countershaft with the tight and loose pulleys for driving machine, but should be higher above machine than shown in drawing.

In treating ore directly from the stamp, too much water may possibly be used by the stamps for proper treatment of the sand by the machine. In such a case there should be a box between the stamps and the concentrator, from which the sand with the

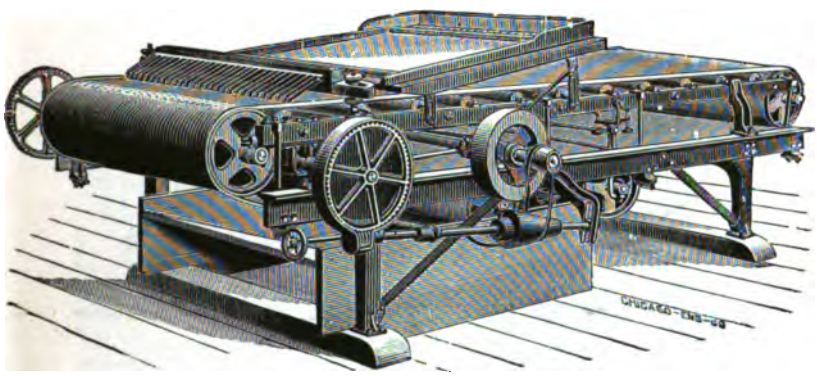


FIG. 59. EMBREY CONCENTRATOR, WITH IRON FRAME.

proper amount of water can be drawn from the bottom; so that the superfluous water will pass away from the top of the box.

The Embrey concentrator,* Fig. 59, is similar in most respects to the Frue vanner, differing chiefly in the direction of the shake, which is longitudinal. Not many of them have been put in use in California, as the Frue Company owns the Embrey patent, and believing the vanner to be the best machine, they have not pushed the Embrey.

Two forms of the Embrey machine are made, one compact in form, with crankshaft passing under the shaking frame, and supporting, by the latter, the whole weight of the belt and two

* Report of the State Mineralogist of California. Part II., page 111.

underhanging rollers. This form is, in fact, identical with the Frue vanner, except that the crankshaft is placed at right angles to the length of the belt, instead of parallel to same, and it is made with iron or wood frame, as preferred.

The second form, and the one most desirable for economy in running, is arranged with the driving shaft at the lower end of shafting frame, so that while running all parts can be regulated, and any play taken up, and the shaking frame does not carry the hanging rollers, which are supported by the main frame, and carry about one-half of the main belt. The shaking frame and mechanism is thus relieved of the weight of the two lower rollers, and nearly one-half of the weight of the heavy belt, an important point, considering the high speed necessary on the end-shake machine.

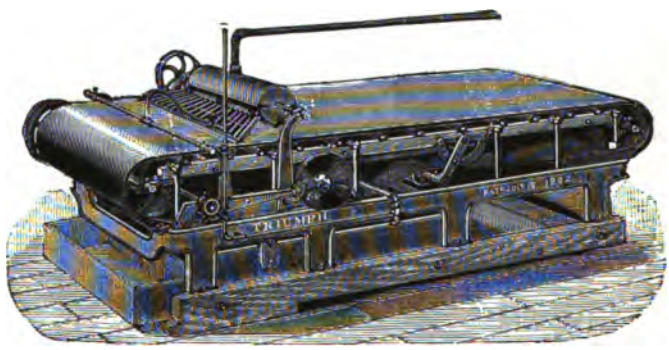


FIG. 60. TRIUMPH CONCENTRATOR.

The Triumph concentrator has an inclined rubber belt, shaking longitudinally, and revolving slowly. Its crankshaft, like the Embrey, revolves from 235 to 240 times a minute, in order to make the separation of the mineral from the sand. Its main novelty is the use of a friction roller to regulate the forward motion of the belt, instead of a cone pulley, which is used in the vanner and Embrey machines. It has also an amalgam saver, consisting of an iron trough in which turns slowly a horizontal shaft, with stirrers. Quicksilver is kept in this trough. The shaking table is supported on upright springs, and the main frame is of cast iron. It can do good work. It is in use at the Empire Mill, Grass Valley.

III. The Grass Valley Method.

The Idaho Mill in Grass Valley, California, Fig. 61, is one of the most successful of all the mines in California, it having been located in 1864 and paid dividends almost without interruption since 1869. It is an excellent example of a mill in which most of the gold is free,* while the Providence Mine in Nevada City, four miles distant, exhibits the other extreme of heavily sulphuretted ores. Steam was formerly used both in mining and milling, but water from the South Yuba ditch under a pressure of 542 ft. has now been substituted for it. The saving is estimated as at least $30,000 per year. In the mine and mill there are now twelve Pelton wheels, three of 8 ft., four of 6 ft., one of 5 ft., one of 3 ft., two of 18 in., and one of 9 in., which together use 320 in. of water with a pressure of 550 ft. Of the free gold 75 per cent. is saved, and 95 per cent. of that contained in the sulphurets, of which the ore contains one-half per cent. These are worth $80 per ton. The sulphurets are first amalgamated raw and then sold to the chlorination works. The value of the ore has varied from year to year. In 1882 it was $20.64 and the cost of mining and milling was $9.83. In 1883 it was $12.76 and the cost of mining and milling was $8.70. In 1884 it was $18.04 and the cost of mining and milling $7.86. In 1886 the cost of mining and milling was $7.65, and in 1887 $8.44. There are in the mill twenty-one blanket sluices, eight Atwood amalgamators, one of which is in reserve, seven Eureka rubbers, four Knox pans, and two buddle concentrators; 35 miners' inches of water are used in the mill. The mill has fifty stamps, but only thirty-five are used. It formerly used sixty-five drops of 10 in. and crushed 91 tons in twenty-four hours through a No. 5 screen. In 1883 this was increased to seventy drops of 9 in. In 1885 to seventy-two drops of 9 in. The mill then crushed 2.85 tons in twenty-four hours per stamp through a No. 5 screen. It now crushes 100 tons in twenty-four hours. The stamps are fed by the Challenge feeder. The stamps weigh 850 lb. and make seventy-two drops of 9 in. The boss weighs 200 lb., the tappet 160 lb., the stem 350 lb., the shoe 140 lb. The shoes

* Stated to contain but one-twentieth to one-half of 1 per cent. of sulphurets.

last 140 days. No amalgamation is done in the battery. The mill is run by a 6-ft. Pelton wheel, which requires 30 in. of water with a $1\frac{1}{16}$-in. nozzle under a pressure of 550 ft. The labour of the mill consists of one man at each Blake crusher, who works a shift of ten hours, and handles 25 tons of rock. For three batteries of five stamps, there was formerly one feeder, who worked a shift of eight hours; he handled 12 tons of rock, and picked out the iron and wood which was in the ore. With the automatic feeder there is but one man for all the stamps.

The lower holes of the screens are 3 in. above the discharge when new. The pulp is discharged from the battery *a*, Fig. 61, into a splash-box *b*, which runs the whole length

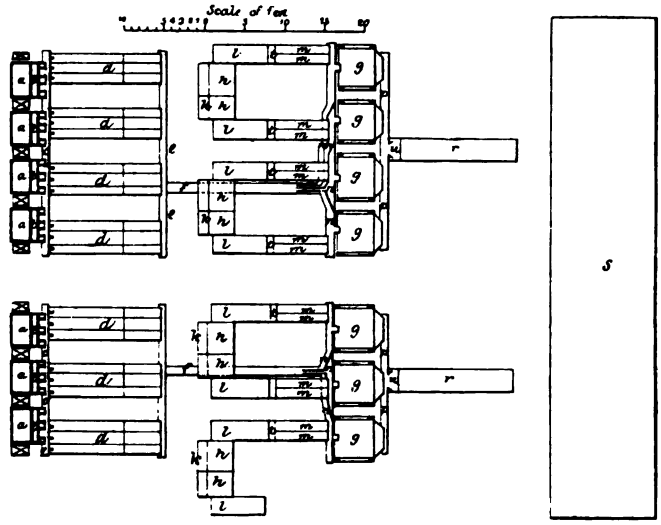


FIG. 61. THE IDAHO MILL.

of the mortar. This box is 9 in. wide on the bottom. The back is the whole height of the screen; the front is only 6 in. high, and is closed along the front, having three openings 6 in. long and 3 in. high, equally dividing the length of the box, which discharge into the trough *c*, which distributes the pulp over the blanket sluices *d*, of which there are three for each battery of five stamps. These sluices are made of $1\frac{1}{4}$-in. planed boards, 16 in. wide. The sides are $3\frac{1}{4}$ in. wide, and are nailed on to the edges of the bottom so as to make the sluice 2 in. deep in the clear. Each of the sluices is made in two parts;

the upper one is 9 ft. 6 in. long, and the lower 4 ft. 6 in.; between the two is a drop of 3 in., and at their end a drop of 6 in. Two of these only are worked at a time, while the third is being cleaned up.

The sluices are so laid that while having a regular pitch of 3 deg. to 4 deg. longitudinally they are made perfectly level transversely. This is necessary to insure the same amount of flow over all parts of the sluice. The grade given to blanket sluices is usually from ¾ in. to 1 in. per foot, depending on the specific gravity and the fineness of the ore and the proportion of sulphurets contained in it. Sometimes the grade is increased to as much as 3 in. to the foot. The apron used in gold mills has always more grade than the sluices, and in many mills there is an arrangement by which the end of the apron can be raised or lowered as different peculiarities of the ore may demand. At their ends the sluices project about 1½ in. over a trough *e*, 6 in. wide, which connects with another *f*, 5 ft. long, there being one for each set of two batteries on one side of the mill and four on the other, which is 18 ft. long and 13 in. wide. These are filled with copper plates and discharge upon a Eureka rubber, which discharges upon a copper plate sluice *r*, 16 ft. long and 32 in. wide. The first sluices *d* are designed to receive the blankets, and have a grade of ¾ in. to the foot, and upon them the battery discharges its pulp. The blankets are manufactured for the purpose, and are 6 ft. long and 20 in. wide, and weigh 1 lb. 10 oz. each. They are made of coarse grey wool, and have the nap on one side only; the other side, which is placed against the bottom board of the sluice, is carefully shorn. They are spread smoothly over the bottom of the sluice, and are made to overlap each other in such a way as to prevent the pulp getting underneath them. The upper sluice receives two of these blankets, the lower one being placed first, and the upper one lapping over it. The lower sluice has but one blanket. The object of the blanket is to arrest the fine particles of gold and sulphurets containing it, which, as they are at the bottom of the shallow stream flowing over them, are caught in the fine fibres of the nap and retained there while the lighter particles of rock pass on and are carried off by the current. The idea of using them was suggested by the

use of hides and fleeces, which were formerly the only materials which could be used for the purpose. These, however, have been completely superseded by blankets. The blankets wear out at the rate of about 1¼ yards per stamp per month; they cost from $1.50 to $2 per yard at the mill. At the end of these blanket sluices there is a passage 30 in. in width, and in front are two tubs *h*, made of 1½-in. boards, 3 ft. 6 in. wide at the top and 2 ft. 6 in. at the bottom, and 30 in. deep, one 3 ft. and the other 4 ft. long. They are provided with a series of plug-holes for discharging the water gradually. Each one has a shelf *k*, Fig. 61, and *a*, Fig. 62, about 8 in. wide, running the whole

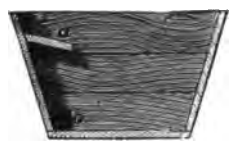


FIG. 62. WASHING VAT.

length of the box. This shelf is sometimes placed outside of the box, where it is not very convenient. It is generally placed inside. The vats *h*, Figs. 61 and 62, of which there are eight, are used alternately.

The pulp from the batteries runs over the blankets, and the gold and gold-bearing pulp in fine particles becomes entrapped in the nap. It is evident that if this process was not interrupted, the lower part of the blanket would become filled with coarse sand, and then nothing would be caught. Besides which, the fine hairs of the nap would become so loaded that they would be pressed down permanently and so destroy the efficiency of the blanket; they are, therefore, only allowed to become full, as after this time they cease to act. The blankets are for this reason washed at stated intervals. The fine particles they contain are caught and settled, and the blankets replaced. In order to secure not only the greater use but the greatest efficiency of the blankets the sluices are made double, as here three sluices are used for two batteries, so that the pulp may be turned into the middle one, while either of the others is being worked, which requires to be done every fifteen to twenty minutes. As soon as one set of blankets have been washed they are replaced, and another set washed, so

that there shall be no interruption to the working of the batteries. At the Eureka Mill there were four sluices for every two batteries of five stamps. As soon as one blanket becomes charged, the pulp is turned off from it, and the blankets taken up and washed in the vat. In taking up the blanket it is carefully folded together, the lower end being carried to the top of the sluice, so as not to lose any of its contents. As soon as the contents of each one is washed out, it is carefully folded together and laid on the shelf in the washing trough, until it is to be used again. The water used in washing the blankets is always warmed by the exhaust steam. The washing tanks are placed in the most convenient position possible, as this is the principal item of labour in the mill. The blankets on the upper sluice catch by far the largest quantity of the gold and pyrites, and are washed every fifteen to twenty minutes; the lower ones catch but little, and are generally washed not oftener than once in two hours. In some mills a stream of clear water is turned into the trough at the head of the blankets in order to dilute the pulp, so as to allow the gold and pyrites to settle more readily, but the more water there is introduced here, the less the inclination of the sluice should be. The quantity of pulp passing is arranged so as to be from 0.5 to 0.7 cubic feet per stamp per minute, which will carry 3 lb. to 10 lb. of solid matter to a cubic foot of water. The blanket concentrates amount to $2\frac{1}{2}$ per cent. of the original ore and have a specific gravity of 3 to 4. There is one man washing the blankets for every three batteries of five stamps each. He watches and regulates the discharge of the screens, sees that the proper supply of water goes over the blankets, charges the blanket washings into the Atwood amalgamator, regulates the flow of the water over them, and works an eight-hour shift.

The quantity of water used in the mill depends upon the amount of sulphurets, or other heavy material in the ore. It averages generally half a cubic foot per minute for each stamp. It must be sufficiently large or rapid to move the crushed sands over the blankets without allowing them to settle, but sufficiently slow to allow the fine particles to be caught in the nap. Less grade of sluice and more water are preferable to heavy grade and a small quantity of water. The total amount of water used in

the mill is from 35 to 40 miners' inches. The water is paid for at 20 cents per inch in twenty-four hours. As each stamp crushes 2.6 tons there will be 4.3 tons of crushed ore passing each blanket in twenty-four hours. The amount collected on the blankets will be about 12 per cent. of the rock crushed, depending upon the relative proportion of the iron pyrites and gold or other metallic contents of the ore. The iron comes partly from the ore, and partly from the wear of the different parts of the stamp.

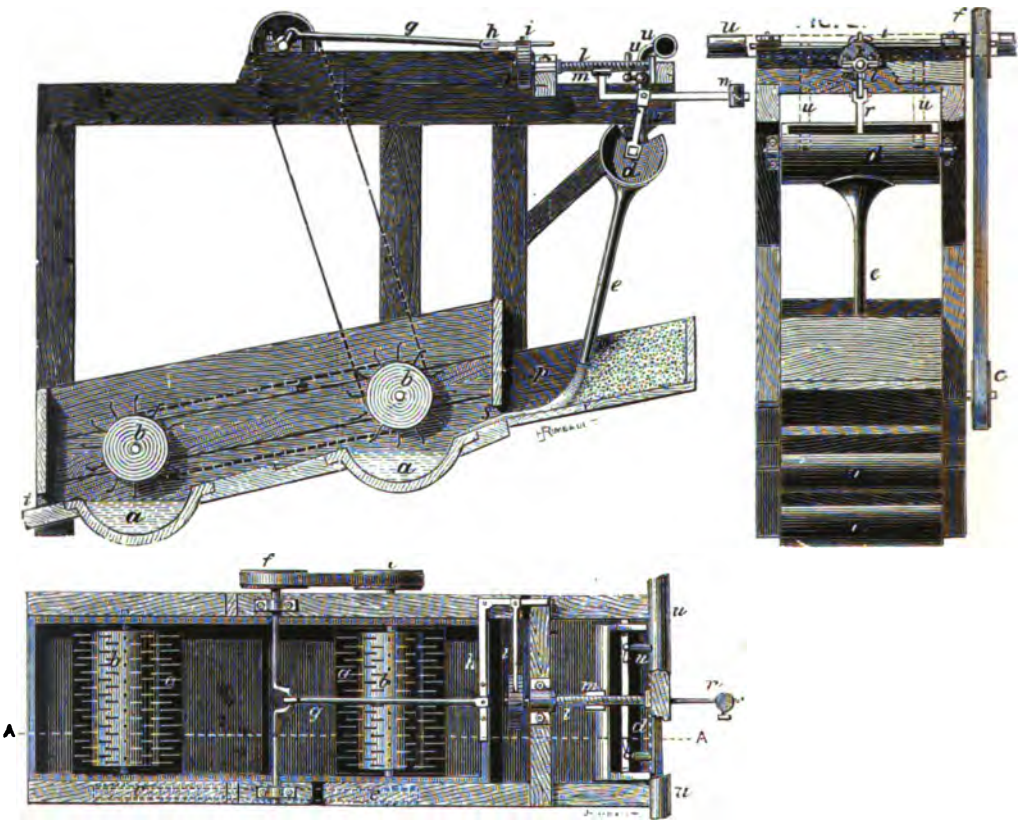


FIG. 63. ATWOOD'S AMALGAMATOR.

From the constant washing of the blankets a considerable amount of material collects in the bottom of the vats *h*, Figs. 61 and 62, which are kept about two-thirds full of water. When the solid material has accumulated so that the vat is too full to do further service, it is left to settle, and the tank on the other side is used. As it settles, the water is drawn off from the top,

and when the settlings are sufficiently consistent to be handled with a shovel, they are charged into the Atwood amalgamator feeder *p*, Fig. 64. This exceedingly simple machine is shown in section in Fig. 63, and in perspective in Fig. 64. It consists of two wooden troughs *a*, 14 in. wide, 6 in. deep, and 19 in. long, which are semicircular in shape, holding from 350 lb. to 400 lb. of mercury each. Over the mercury a cylinder of wood *b*, 8 in. in diameter and 18 in. long, is made to revolve. The circumference of this cylinder is divided into twelve equal parts, and its length into eighteen. Commencing at each end with half a

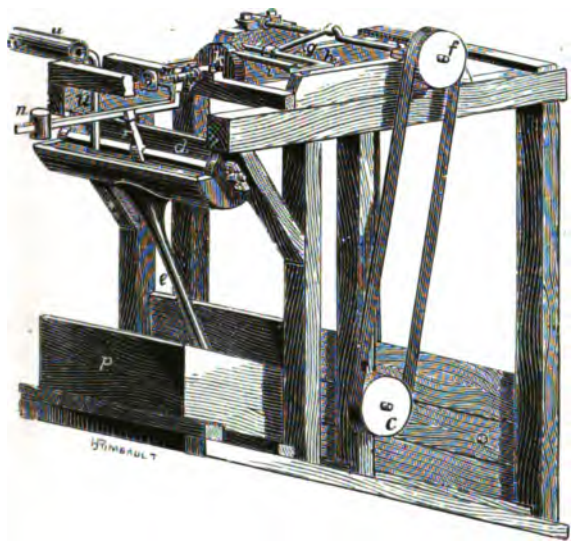


FIG. 64. ATWOOD'S AMALGAMATOR.

division, iron rods ¼ in. in diameter and 3 in. long, are set at right angles to the surface of the cylinder, but slightly curved at one end. They are set in quincunx, but do not touch the mercury although they come very close to it. There are two of these cylinders set 3 ft. apart. The lower one is 6 in. below the upper, and between them there is a riffle. The box containing them is 4 ft. long, 21 in. wide, 14 in. deep on the outside, and has an incline of 3 in. to the foot. The two cylinders are moved by a belt running over two pulleys and making sixty revolutions per minute.

When the washings are sufficiently dry to be handled, they

are taken out of the vat with a shovel and placed in a charging box *p*. As it is necessary that the pulp should be made to flow with the greatest possible regularity through the amalgamator, an ingenious automatic feeder is arranged on the top frame of the amalgamator. The wheels *c* and *f* communicate motion to an arm *g*, which communicates with an iron lever *h*, bent at right angles, and hinged at the angle. The end of this arm to which *g* is attached, has three holes to admit of a variation in movement. On the other end of *h*, three holes are also made to admit of a change of movement there, and to one of them a ratchet *i*, moving on the ratchet wheel *k*, is attached. This wheel is made to move forward one ratchet for every revolution of *c* and *f*. Keyed on to *k*, and serving as an axis of rotation, is an iron bar *l*, with a screw thread cut on its whole length, against the under side of which a half nut *m*, kept in position by the counterpoise *n*, is placed. To the counterpoise lever the water feed-box *d* is attached. By raising the counterpoise arm the pipe *e* can be placed in any position in the box *p*. The screw and ratchet are regulated so as to move just fast enough to wash the pulp out of the feed-box.

The blanket washings are charged in the back of the feed-box *p*, which is 18 in. wide, 3 ft. 3 in. long, and 10 in. high. The pipe *e* is set so that the water will flow over the ore, and wash it out into the amalgamator; it moves backwards as fast as the ore is washed out. The water used for this purpose is heated to a temperature of 120 deg. to 130 deg. Fahr., as it is found that the amalgamator works better hot than cold. The washings pass through the first amalgamator, over the riffles, on to the second, and out at the discharge *t*. The heavy particles, as they pass under the wheels and over the riffles, will sink to the bottom. The gold, except that contained in the pyrites, or attached to small particles of ore, will be caught by the mercury of the basin or riffles, while the constant agitation of the rods on the wheels *b* will tend to keep the lighter particles of poor material in suspension. As the material which passes through this machine is the most valuable part of the ore, great care is taken not only that the pulp shall be evenly and constantly fed, but also that the surface of the mercury is kept bright. Both that in the basins under the wheels and in the riffles is continually watched,

and cleaned as soon as its surface is no longer quite bright. These skimmings are put on one side for after treatment. The success of the mill depends in a great measure on the care with which this work is performed. The automatic feeder does not need much watching. The water feed-pipe *e* occasionally gets stopped or impeded in its motion, but the mercury requires constant attention. Most of the gold in the mill is caught in this machine.

The blanket concentrates, after passing the Atwood amalgamator, pass with the battery pulp into a box *t*, Fig. 61, 1 ft. long and 21 in. wide, from which the stream is divided into two parts to be distributed over two sluices *m*, with corrugated amalgamated copper plates. These are made to oscillate so as to agitate the sand. These sluices are 8 ft. long, the two being 21 in. wide, with a grade of 1 in. to the foot. From here they discharge upon the Eureka rubbers *g*, Figs. 61 and 65, of which there is one for every battery of five stamps. The water from the battery discharging at the temperature of the air into the rubber, meets here the warm water from the amalgamators, so that the temperature is slightly raised. The object of the Eureka rubber is to clean any rusty gold; to rub off by friction any slime which may have attached itself to it, or to brighten the gold, so that it may more easily amalgamate. It is made of a cast-iron box *a*, Fig. 65, 1½ in. thick, and 56 in. square and 7 in. deep, into which a false bottom, 2½ in. thick, is placed. This false bottom is made up of three parts; the dies *c*, made of soft wood, cut across the grain, which are 4 in. by 11 in. and between which the wooden die bedplate *b*, 3½ in. wide and 1 in. thick, and the whole width of the box, is placed. On the top of this die bedplate, the cast-iron dies *d*, which are 3½ in. wide at the bottom, 4 in. at the top, and 2 in. thick, are placed; the object of the flare being to hold the wooden dies *c* in place when the surface begins to wear. The bottom of the box thus becomes a die, consisting of alternate strips of wood and iron, which are held firmly in their place by wooden strips *e*, 1 in. thick, let ¾ in. into the dies, and bolted to the side of the box. Over the die a wooden frame *i*, held together by two bolts *s*, is suspended from the top of the frame *p* by iron rods ½ in. in diameter. It is moved backwards and forwards by

means of an eccentric *n*. To the bottom of this frame cast-iron footplates *j*, having six slots formed by flanges projecting 1½ in.

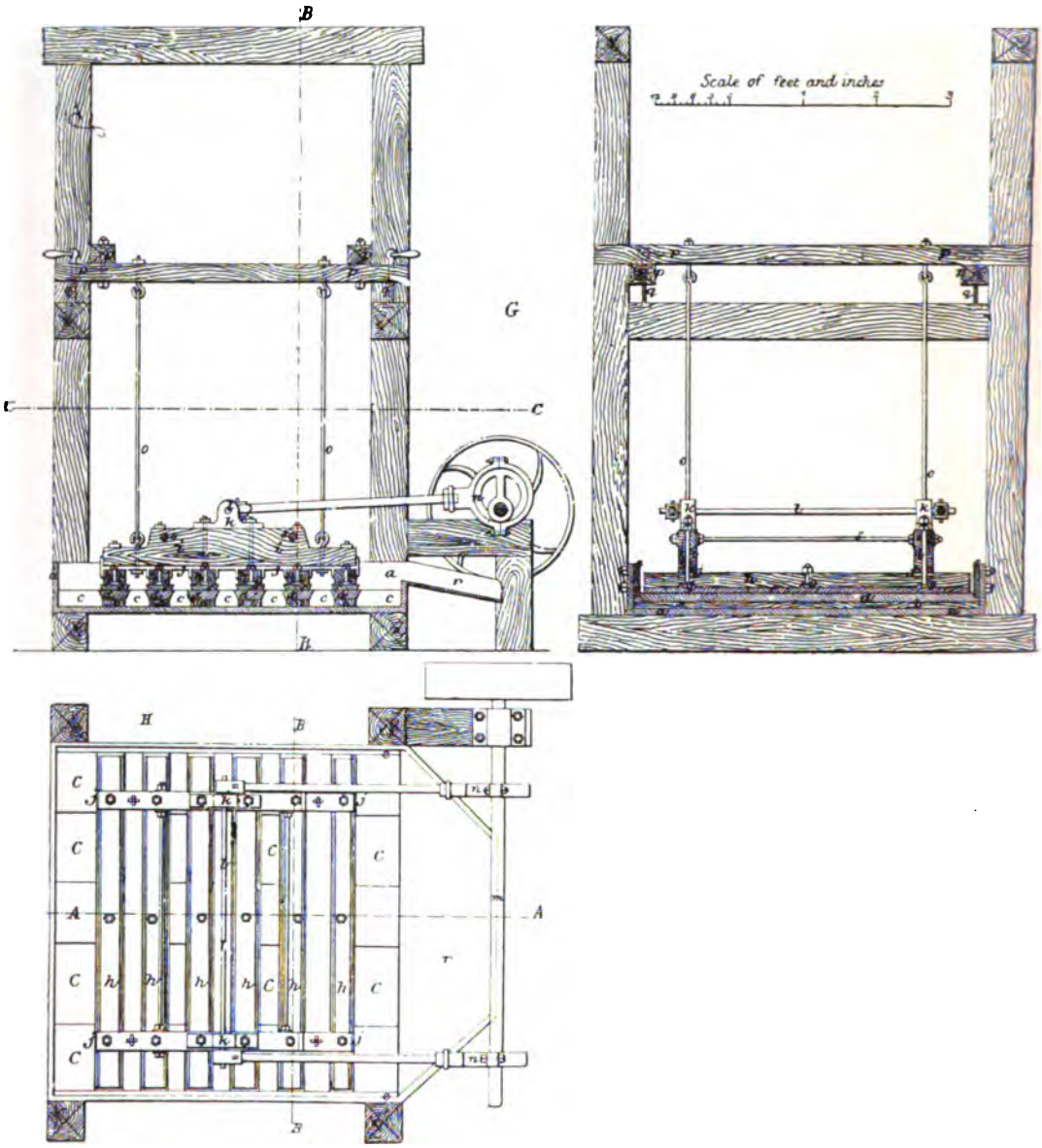


FIG. 65. EUREKA RUBBER.

to receive the wooden shoes, are placed. The wooden shoes *h* are 3½ in. wide and 2½ in. thick, and have iron shoes *f* 4 in. wide

at the top and 3½ in. at the bottom, and 1 in. thick, fastened to them by means of sockets *g*, Fig. 66, cast on them. Both the iron and wooden shoes *f h* have amalgamated copper plates attached to them. The arrangement of the shoes and dies is clearly seen in the perspective drawing, Fig. 66. The rods *o* suspend the shoe frame from a movable frame *p* which is capable of adjustment by means of a screw *q*, which allows the shoes and dies to be set at any distance. The eccentric which communicates the movement to the shaft *l*, which goes across the top of the frame, makes fifty-five revolutions per minute, and gives a motion of 4 in. backwards and forwards.

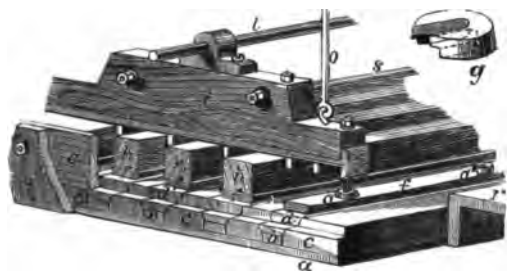


FIG. 66. SHOE AND DIE OF THE EUREKA RUBBER.

The pulp from the Atwood amalgamator is discharged from a trough *n*, Fig. 61, 9 in. wide, so arranged as to give an equal supply to each rubber; passing into the rubbers, it goes under the shoes and dies, and the bright surfaces are caught by the amalgamated plates. From the rubber it is discharged over the apron *r*, Fig. 65, the sides of which converge to an opening 27 in. in width, and into a trough 6 in. wide, communicating with three rubbers on one side and four on the other, on to another head-box *n*, Fig. 61, by four holes 2 in. in diameter. Some of these holes are closed to make the flow on to the plates even. From the head-box the pulp flows through boards with four rows of ½-in. holes. The rows contain twenty and twenty-one holes respectively, and leading to a sluice *u*, made of 1-in. plank 15 in. long and 32 in. wide, from which there is a drop of 3 in., where it discharges into another sluice *v* of the same width, but 16½ ft. long, with a grade of half an inch to the foot, in which the water is discharged in intermittent waves. The sides of this sluice are 3 in. high, and the sluice is filled with amalgamated plates

30 in. wide. There were formerly only two or three, but there are now seven of them, one to each battery. They are covered with three-lapped copper plates in each sluice. These plates are dressed, without stopping, once a day, by sprinkling a little mercury on them. They are cleaned twice a week. These discharge by gratings into a sluice which delivers to the tail-house. From here the tailings are discharged into settling vats to be concentrated. In the Father de Smet Mill the Eureka rubbers were placed at the end of the amalgamated plate sluices; they were, however, subsequently taken out. The quantity of material employed for the battery was too large for the rubber, and the grinding caused a loss of mercury.

A certain amount of skimmings, which varies between $\frac{1}{2}$ and $\frac{1}{10}$ per cent. of the battery sands, is taken from the amalgamators and the riffles which are placed in the sluices in front of them. The skimmings are ground and amalgamated in four Knox pans, Fig. 67. This pan is one of the oldest and most extensively used of all the pans invented to clean gold amalgam. It is made of cast iron, and consists of a pan 50 in. in diameter at the top, 47 in. at the bottom, and 14 in. deep. In the centre a hollow cone 13 in. in diameter at the bottom, 6 in. at the top, and 17 in. high, forms part of the casting of the pan. Into the annular space formed by the cone and the sides of the pan, a false bottom *a*, of cast iron $1\frac{1}{4}$ in. thick, with projections on the under side $\frac{1}{4}$ in. high and thick on both diameters, are cast, to serve as a steam chamber *b*. This false bottom fits the sides of the pan tight. On the top side there is a groove *n*, $1\frac{1}{8}$ in. wide and 1 in. deep, from the centre to the circumference, communicating with a lower discharge hole *s*, which is at right angles to the shaft *i*. A cast-iron muller *e*, made of four arms each 13 in. long, $4\frac{1}{2}$ in. wide, and 1 in. thick, is attached to a circular piece $4\frac{1}{2}$ in. wide and 19 in. in diameter. Each arm has a slot *g*, 10 in. long and 1 in. wide, to receive the bolts of the shoe, and allow of freedom of motion. On the central rim there are two projections *o*, with short arms *p*, for the purpose of receiving the yoke *f*, which is keyed to the shaft *j*, which is 2 in. in diameter, and is guided by the top of the hollow cone, and the plate *r* screwed to the bottom of the pan. The projection *o* forces the muller to revolve

in one direction; when turned in the opposite one, the yoke *f* frees itself. At the end of each arm *e* the shoes are attached by a bolt which moves in the slot *g*. The shoes consist of two parts

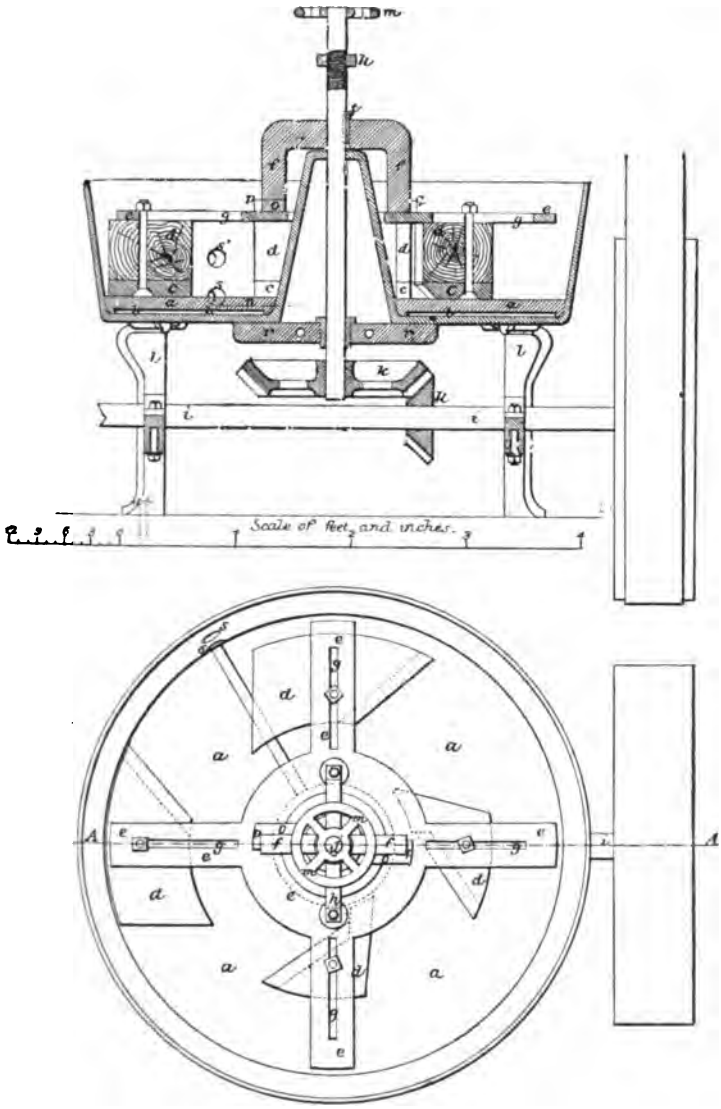


FIG. 67. THE KNOX PAN.

which have exactly the same shape. The under part or shoe proper *c* is of iron 1½ in. to 2 in. thick. Above this there is a wooden shoe *d*, 6 in. in thickness, sufficiently high to rise above the

pulp in the pan, the only object of which is to prevent the pulp from settling and remaining on the iron shoe. Sometimes the shoes are made entirely of wood. Two of the shoes are made to run on the outside and two on the inside, the shoes lapping on each other, so as to more than cover the surface of the bottom. At right angles to the movable yoke *f* there is another one *h*, which is screwed to the muller. It is fitted to the shaft by means of a screw thread cut on it. The wheel *m* serves to raise or lower the muller. The whole pan rests upon cast-iron legs *l*, which are slipped into grooves like those of an ordinary house stove. The movement is communicated by a horizontal shaft *i*, and two bevel wheels *k*, which cause the muller to revolve at the rate of fifteen to twenty revolutions a minute.

In the sides of the pan there are two discharge openings *s s*', which are 2 in. in diameter, and are closed by wooden plugs. The lower one is at the end of the radial groove *n*, in the top of the muller, and the other is 4 in. above it. Sometimes for very small mills the shoes are made entirely of iron, the bottom is then made very heavy so that iron surfaces come together. Such pans are much used in working up small quantities of concentrates from blanket sluices, when the attrition of the surfaces can brighten the gold and put it into such a condition that the mercury can attack it. Sometimes the pans are made with removable grinding surfaces so that they may be used with any of the three varieties of shoes. The weight of these pans is given below :

	lb.
4-ft. clean-up pan weighs	3000
3-ft. ,, ,, ,, ...	2000
30-in. ,, ,, ,, ...	1500

When they are to be used in mountainous countries they are made in sections so that they can be packed on mules. Each piece then weighs only 300 lb. or less.

The pan requires very little repair. The false bottom wears out in about a year, when it can be replaced independently of the rest of the pan. The wooden shoe lasts three months; the shoes and dies from three to four weeks. The charge of skimmings introduced is variable, generally from 100 lb. to 125 lb. Water is then added, until the pulp is sufficiently strong to attach

itself to a stick without dropping from it; 10 lb. to 15 lb. of mercury are then added, and the pulp ground for three hours with the muller down. At the end of this time 5 lb. or 6 lb. more of mercury are added, and about a gill of equal parts of the concentrated solution of saltpetre and sal-ammoniac, and sometimes some mercuric nitrate and copper sulphate. Steam is now introduced under the false bottom, into the space *b*, and the arms turned with the muller slightly raised for three hours. At the end of this time the amalgamation is finished. The pulp is diluted, so that the water rises nearly to the top of the pan. About a dipperful of quicklime in small quantities at a time is now added, as it is supposed to assist in collecting the floured mercury, and after this the muller is kept constantly revolving for half an hour, when, it being still in motion, the upper discharge is opened, and the charge allowed to flow into the tail sluice. The pans are flooded in this way twice a day to wash out the tails. They are cleaned up only twice a week. In order to catch any mercury or amalgam which may be in fine particles and carried off, the stream is made to fall into a pail, from which it overflows and runs to the settler. When the upper hole has discharged the one below is opened. The discharge from the pan caught here contains from $50 to $125 to the ton; most of it is sulphurets. The settlings are saved to be treated with the other sulphurets.

When the water is entirely out, the remaining contents of the pan are washed into a pail; mercury is then added to it, and water, and the whole stirred, and the skimmings put back into the pan. The resulting amalgam is treated in the same way as the rest of the amalgam in the mill. The amount of mercury used in the mill is about 130 lb. per stamp. The loss is about 1½ per cent.

One amalgamator, who works a ten-hour shift, is all the labour required. He attends to the screens, cleans the amalgam from the plates and worn-out shoes and dies. Besides him, there are two engineers on twelve-hour shifts, and one night watchman. All the men in the mill are paid $3 per shift, the length of the shift depending on the work they have to do. Besides these there is a foreman, generally paid $5 a day. The total cost of

milling a ton in 1874 was $2.22. The cost of mining was $6; total cost of mining and milling, $8.22.

To clean the Atwood amalgamator, all the pulp floating on the surface of the mercury in the troughs is first removed. The mercury is then carefully taken off at the top, as the amalgam is heavy, and lies on the bottom of the trough. When the mercury is all removed the amalgam is taken up separately. The mercury from the top is then replaced in the troughs; it contains a little gold which is usually not separated, as it will all be caught in the subsequent clean-ups. From 90 to 95 per cent. of the amalgam is found in the upper trough. The amalgam riffle and the riffles outside of it are cleaned in the same way. The amalgam is collected with an iron tool bent to fit the riffle, and is added to the other amalgam. The skimmings are all sent to the Knox pans. The amalgamated plates are cleaned with a chisel and scraper, care being taken not to damage the surface of the copper. The plates are then rubbed and re-amalgamated. The amalgam is treated in the mortar with a large addition of mercury, as the lumps are quite hard. The amalgam when strained will contain from 35 to 40 per cent. of precious metal, the quantity of gold depending upon the amount of silver in the ore. It contains besides some iron, lead and copper which have been amalgamated. To clean it and free it as far as possible from these metals, a very large amount of fresh mercury is added to it, and it is worked over either by hand or in a pan made for the purpose, the object of which is to break up the lumps and thoroughly incorporate the whole in mercury. The impurities rising to the top are carefully skimmed off and put on one side for further treatment. When the surface of the mercury is bright it is strained in a canvas bag, or, if the quantity is small, in a piece of buckskin, the amalgam remaining behind being squeezed and rubbed by hand while in the bag or buckskin, to get out as much mercury as possible. When dry, that is free from extra mercury, it is quite plastic, and is made up into any convenient form for the retort, the pieces used generally weighing from 30 oz. to 40 oz.

According to Mr. Van Deetken the gold from the mill after

leaving the stamp is found in the following proportion in the different parts of the mill ;

Atwood's amalgamator	65.5
Knox pan	26.
Eureka rubber	4.5
Copper plates	2.
Riffles	2.
Total	100.0

The battery gold varies from 10 to 20 per cent. of the total yield, and the concentrators give 5 to 10 per cent. The gold obtained from the whole process will be distributed as follows :

Battery	9.5
Mill	85.75
Concentrators	4.75
Total	100.00

In the case cited the yield of the mill was $47.40 ; the concentrators $2.50 ; and the value of the tailings was $16.49 ; of the float gold, $2.28 ; total $68.67 ; 27 per cent. of this is lost, 22⅔ per cent. as tailings, and 3⅓ per cent. as float gold.

The following Table gives the details of the cost of crushing and amalgamating one ton at the Idaho Mill in the year 1874 :

	$	
Supplies.		
Quicksilver	.0350	
Blankets	.0109	
Screens	.0020	
Chemicals	.0059	
Hardware and light	.1123	
Lubricating material	.0078	
Charcoal	.0022	
Hauling of freight	.0047	
	——	.1808
Labour.		
Rock breaking	.1365	
Feeding	.2494	
Blanket washing	.2494	
Night watch, half shift	.0220	
Amalgamator	.0531	
	——	.7104

Brought forward		.8912
Management.		
Foreman, half salary	.0435	
Manager ,, ...	.0871	
	——	.1306
Repairing.		
Carpenter and blacksmith	.0108	
Lumber	.0034	
	——	.0142
Steam Power.		
Engineer	.1210	
Wood-wheeling boiler	.0417	
Wood	.3963	
	——	.5590
Sundries.		
Water rent...	.0949	
Foundry	.2676	
Interest on cost of mill, at per cent.	.1749	
Payment toward repayment of capital invested, per ton	.0876	
	——	.6250
		$2.2200

The following Table gives the distribution of the total yield from the year 1871 to 1874:

——	1871.	1872.	1873.	1874.
	$	$	$	$
Bullion	385,017.90	390,830.50	979,959.89	631,190.56
Sulphurets . . .	10,041.23	8,872.10	31,318.00	21,600.00
Tailings	...	...	12,950.00	10,989.25
Gold in specimens sold .	296.50	762.73	364.50	38.50
,, from old copper. .	...	...	...	929.89
Total yield in gold .	395,355.63	400,465.42	1,024,591.89	664,811.20
Average yield per ton .	35.50	35.09	37.91	23.40
Weight of Bullion and Sulphurets.				
Bullion	...	22.331.46 oz.	56,210.80 oz.	36,169 oz.
Sulphurets . . .	92 tons	83.75 tons	212.5 tons	217 tons
Total number of tons treated	11,133 ,,	11,410 ,,	27,625 ,,	28,801 ,,

After it leaves the mill the pulp becomes tailings, which are subjected to further treatment, having in view the saving of as much free gold as possible, but principally the concentration of the auriferous pyrites, so that it can be afterwards subjected to chlorination. This treatment was formerly effected at the Empire Mill in Grass Valley, as follows: The sands (*a*) are first

sized, (*b*) then concentrated, (*c*) and afterwards tossed. The particular methods of treatment are peculiar to Grass Valley.

The following Table, extracted from the reports of the company, gives the work done with the yield, cost, and profit from the year 1870 to 1874:

—	Amount Crushed.	Yield.	Yield per Ton.	Cost of Mining and Milling.	Profit.	Number of Stamps.
	tons.	$	$	$		
1870	9,782	187,034	19.03	11.85	7.15	15
1871	11,333	395,355	35.50	10.20	24.30	15
1872	11,410	400,465	35.09	13.08	22.01	15
1873	27,624	1,024,591	37.91	8.61	29.30	35
1874	28,801	664,811	23.40	8.22	15.18	35
1875	28,013	495,669	17.63	9.06	8.57	35
1879	28,072	440,445	15.42	8.96	6.46	35
1880	27,945	430,837	15.82	9.29	6.53	35
1881	27,509	554,293	22.95	9.51	13.44	35
1882	27,639	627,000	20.64	9.83	10.81	35
1883	28,572	364,599	12.76	8.71	4.06	35
1884	31,143	561,895	18.09	7.86	10.18	35

The sands are sized in pointed boxes, Fig. 68, placed in the bottom of the tail sluice, the number depending upon the other

FIG. 63. THE POINTED BOX.

concentrating appliances used in the mill. The length is always measured in the direction of the current, and depends upon the size of the ore. The shorter the box the larger the grain. No definite rule can be given for making these boxes, except that the depth is made equal to the length; the other dimensions are determined by experience. The dimensions used in the Keystone Mill, Amador County, are:

	in.
Length at top	14
,, bottom	6
Width	10
Depth	14

From the bottom of the box rises an iron pipe, 2 in. in diameter, which is open at the top, and has three or more openings in its height, which are 1½ in. in diameter, the centres being from 4 in. to 6 in. apart. These openings are made of gas-pipe fittings. The quantity of water to be discharged with the sands will be regulated by the position of the discharge, the greatest amount being discharged from the lowest hole; when there is more than one sluice, the tailings passing over the first flow into the second, and so on. What passes the last goes to waste. This concentration is effected by gravity only.

The material flowing from the pointed boxes runs into wooden sluices, 18 ft. long, whose width is determined by the number of the stamps discharging upon them; it is usually 1 in. per stamp for coarse sands and 1¼ in. to 1½ in. for fine. They are generally 15 in. to 16 in. wide. At their end they are provided with automatic riffle gates, raised by a ratchet wheel set on a vertical screw and attached to the gate. The heavy material deposits near the head, while the lighter portions go to waste over the riffles. The gate rises gradually, and once in twenty-four hours is fully up. The opening is then 7 in. wide by 18 in. deep, one gate answering for two sluices. On the vertical screw there are two ratchet wheels with reverse motion, one to raise and the other to lower the gate. Three batteries of five stamps will fill two such boxes, 18 in. deep, in twenty-four hours. There must be two sets of sluices, so that when one is full the other can be used.

At the Idaho Mill the tails from the mill at the end of the plates are run through a grating into four tanks 8 ft. square, in which there are two partitions. The light particles discharge from one of these at the top and the heavy from the bottom. These run into a set of eight sluices on each side, which are 20 ft. long, 2 ft. wide, and 48 in. deep. The discharge gate is raised by a screw, which runs with a ratchet, so that the sluice is emptied in twenty-six hours.

From the sluices the concentrated tailings are treated in Paine and Stevens' concave rotary buddles, Fig. 69, of which there are always two, one for the coarse, and the other for the fine concentrates, or they are worked alternately, one being filled one day to be cleared the next, and so on. They consist of a concave fixed

table *s*, 18 ft. in diameter, with a central ring of lower depth 30 in. in diameter. From the centre of this opening a vertical shaft *a* is carried to the peak of the frame, and is moved by a bevel gear, moved by a belt running over two conical drums to allow of regulating the speed of the motion.

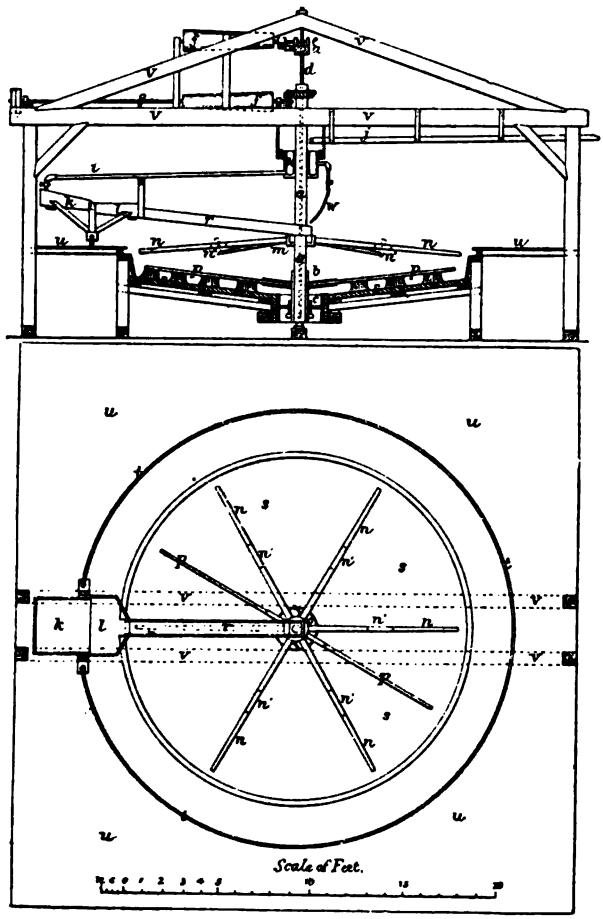


FIG. 69. PAINE AND STEVENS' BUDDLE.

When the tailings are to be treated, the buddle being empty, the top of the riffle gate *c* is lowered to the edge of the fixed floor. The tailings are charged into the charging box *k*, and discharged into the sieve box *l* by means of a stream of water coming through the feed-pipe *i* capable of furnishing at least 8 cubic feet per minute, which is connected with a revolving

cistern *h* moving on wheels, fed by a sluice *j*. The water enters the feed-box and washes out the ore into the sieve-box sieve, whose object is to separate any coarse particles which may get into the tailings; it runs into the sluice *r*, and from these into the feed-box *m*, from which it flows into six hollow arms *n* made of pieces of iron put together, so that they can be separated to be distributed over the floor. The water-box, feed and sieve-boxes, and launder *j*, are all attached to the vertical shaft, but in such a way as to be independent of it. As they rotate both above and below they can be moved to any part of the upper buddle floor *u*. The feed arms *n* are made of three parts. The material discharged from their end falls upon the inclined floor at its upper end, and is carried over it by gravity. The heavy material rests upon the outside, while the lighter discharges over the central riffle *c*, which is made of a ring fitting the side of the central opening, which rises as the ore accumulates, but allows the water and light material to discharge over its edge and through its centre. To insure a perfectly even deposit brooms are attached to the movable arms *p*, which rotate on the vertical shaft with the feeders just touching the surface. They are arranged on a sleeve *b* on the shaft so as to rise with the round riffle as the material deposits. The speed of the buddle is regulated by conical rollers *q*, which are 4 ft. long, so that it can be changed at any time by altering the position of the belt.

It takes twelve hours to fill the buddle with fine sand, and eight hours with coarse. When the buddle is full the circular riffle is lowered; the material is then divided into three parts, called the heads, middles, and tails. The interior ring, 3 ft. 6 in. in diameter, is called the tails, and is worthless. It is washed out by taking off the end of the feed-pipe, and turning it down by an adjustment allowing clear water to flow into the feed-box from the pipe *w* by a rubber hose, and to play on this part only as it revolves. This washes out the tails entirely. A ring 3 ft. in width at the outside, called the heads, is now shovelled out on to the platform *u*, leaving another ring about 2 ft. wide in the centre, called the middles. These middles are rewashed in the buddle, and divided as before, the heads going to the first buddle heads, and so on. The first buddle heads with the

second are washed by raising the round riffle about 3 in., and filling up the buddle floor with tossing-tub tailings. The heads are then charged into the mixing trough, which is pushed to any part of the floor for that purpose. When ready to do this work, the buddle floor has a grade of $1\frac{3}{4}$ in. per foot for coarse and $1\frac{1}{2}$ in. for fine sands. It takes three hours to wash the contents. When they are all washed the buddle is divided as before into three parts, the tails being 4 ft. in diameter, the heads 1 ft. 6 in. wide, and the middles 3 ft. The heads go from here to the tossing-tub, the middles are re-treated as before. Two such buddles require three men at $3 per day. As there is more or less loss of sulphurets from these tailing concentrators, the tails pass over a long set of riffle sluices, which pass through a box 10 ft. long, 5 ft. wide, and 3 ft. deep. The sulphurets caught in these drop boxes are re-treated. With all these arrangements, there is still a loss of 12 to 13 per cent. of the sulphurets.

The tossing-tub or keeve, Fig. 70, is made of wood, and is 4 ft. in diameter at the top, 3 ft. 6 in. at the bottom, and 30 in. deep. It is made of 2-in. staves bound by three iron hoops, $2\frac{1}{2}$ in. wide and $\frac{1}{4}$ in. thick. The staves continue 2 in. below the bottom, from which point four of them continue 10 in. to a platform 6 in. high, to which they are bolted, so that the bottom of the tub is 18 in. from the floor of the mill. Through the centre of the bottom a hollow cast-iron tube *b*, 3 in. in diameter, passes, and is bolted to it by a flange cast on it. Through the centre of this tube a wrought-iron shaft *c*, 2 in. in diameter, moved by the gear wheels *d*, passes, and receives on a key at its upper end the yoke *h* with its eye *m*; to this yoke eight stirrers *k*, of flat iron 2 in. wide and of variable length, are bolted. Attached to the gear wheels *d* are two pins *e*, which trip a rectangular arm *f* with a counterpoise *g*, ending in hammers. When the headings are ready, a wedge of wood is shoved between the hammer handle and the side of the tub, so that the pins *e* do not strike the lever *f*. The tub is filled half full of water, and the stirrers made to revolve forty-eight turns per minute. The tailings are shovelled in near the outside until the tub is nearly full. The stirrers are then lifted out by means of a rope with a hook which passes

through the eye *m*, and the hammers which have been fastened by the wedge loosened so that the pin *e* acts on the lever arm. As the velocity continues the same, these hammers each strike the iron *o* attached to the side of the tub ninety-six to one hundred times per minute. In this way the material in the tub is very quickly settled. When it is settled the water is drawn off from the top, and

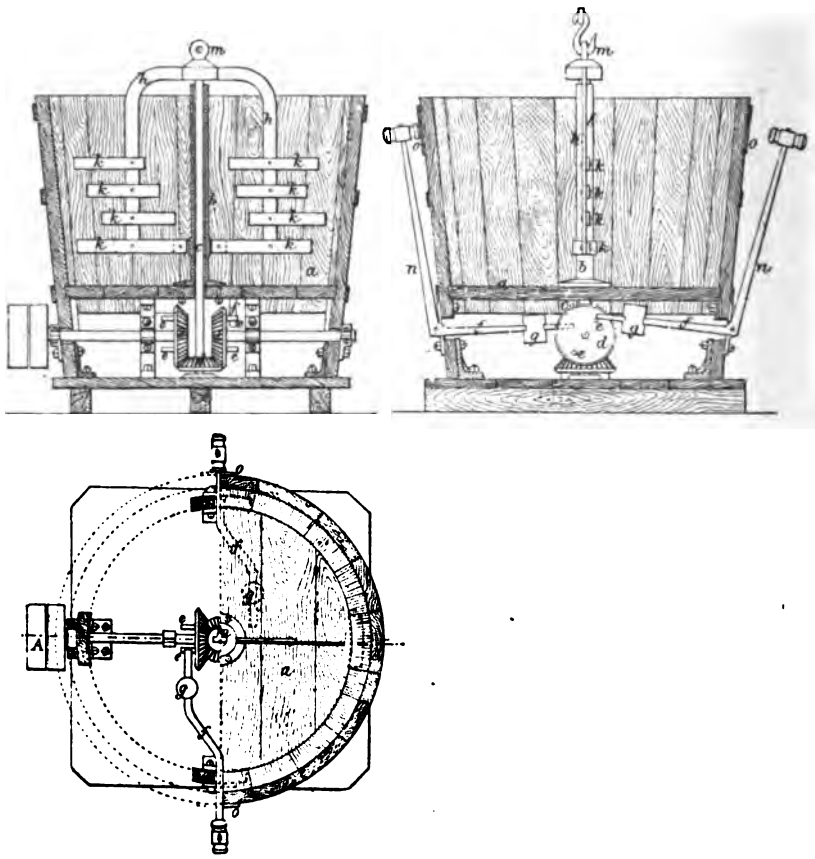
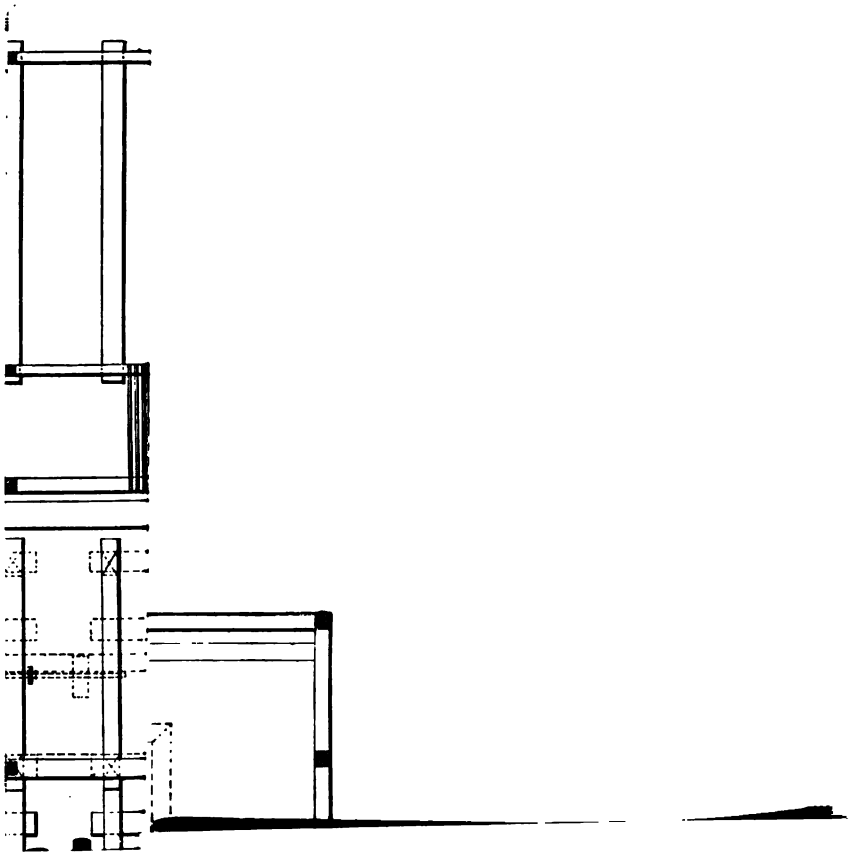


FIG. 70. THE TOSSING-TUB.

the material in the tubs is divided into tops, middles, and bottoms. The tops, which are about 2 in. in thickness, are carried off and thrown away. The remainder is divided into two equal parts; the middles are taken out and re-tossed. The tops from these are thrown away. The middles go to the buddles; and the first and second bottoms are clean sulphurets, and go to the chlorination works. They are worth $80 to the ton, and cost $20 to treat



by that process. The total cost of concentration by this method is from $0.14 to $0.15 per ton of rock stamped.

From the buddles the tails of the Idaho Mill run over a set of four sluices 150 ft. long and 12 in. wide. In this sluice there is one riffle every 16 ft. These riffles are put in one in the morning and one in the evening. The sand is taken up in the morning, and rocked in a rocker 12 ft. long and 24 in. wide. At the end of this set of sluices there is a head-box for clear water. The space is then widened so as to make six sluices 20 in. wide and 200 ft. long. On these there are no riffles. The sand is simply taken up and worked in the rocker. The result is fine sulphurets. From the end of this sluice the tails are carried by a sluice a quarter of a mile long into a house, in which there are sets of sluices 22 in. wide, the bottom of which is covered with duck. Eight sets of these sluices are called "a plant." In one house there are fourteen "plants," 18 ft. long by 22 in. wide, and in the other seven plants, 16 ft. long and 22 in. wide. These sluices become loaded about every half-hour, and they are washed lightly with a broom. This discharges the light sand. The sulphurets collected on the duck are then washed into a collecting sluice by turning over a board at the end of the sluice, which allows all the washings to run into six settling tanks, 16 in. by 4 in. and 36 in. deep. When these are full of water the sand is settled by beating the water with a shovel into waves. The discharge into the sluices is made by gates, which move in a groove. In the first set there is one of 1-in. holes, in the other two of ¾-in. holes. The settled material is re-treated in two "plants" of six sluices, 20 ft. long by 22 in. wide. The concentrates are sold. They are worth from $75 to $300 to the ton. The tails run on to twenty sluices, 30 ft. long by 22 in. wide, on which gunny sacking is placed. These concentrates are ground in a pan run by a Pelton wheel. The mill does all the work up to the quarter of a mile sluice. The work beyond it is done on a royalty.

IV. Battery Amalgamation and Concentration.

The Montana Company's Drumlummon 60-stamp mill at Marysville, Montana, Fig. 71, is an excellent example of a well-appointed establishment erected to treat a low-grade ore. The

company has three mills, a 60-stamp gold mill, a 50-stamp silver mill, and a 10-stamp trial mill. The mill building of the 60-stamp is constructed of wood, which is painted with iron paint. It is 60 ft. high, and is divided into five levels, which are the ore, the breaker, the battery, the plate, and the concentrator levels. A 3-in. iron water pipe runs on the peak of the roof of each compartment. This is provided with automatic fire plugs. The pipes are kept full of water in the summer, but in the winter they have to be drained. Each level is supplied with hose on a reel, suspended in a such a way that the running off of the hose opens the valve and lets the water on. The whole works are lighted by electricity, for which there is a special plant. The ore goes to the mill in mine cars. It is dumped from a counterbalanced car into the supply bins, which hold 2500 tons. The shaft cars hold 1730 lb.; the mule cars 2200 lb. These are all filled automatically in the mine from shoots. The battery bin below the supply bin holds 500 tons, making the total storage capacity of the mill 3000 tons, so that the mill can still work at its full capacity if the mine supply is cut off. From the supply bin the ore goes over grizzlies 6 ft. wide and 12 ft. long, made of $\frac{3}{4}$-in. bars 2 in. wide, set $1\frac{1}{2}$ in. apart at an angle of 54 deg. The large ore from the mine goes to three Blake's crushers, whose jaws are 9 in. wide and 15 in. deep. One of the plates of these crushers, both here and at the Treadwell Mill in Alaska, is set with keys, and the other is leaded with a $\frac{3}{4}$-in. lead plate, which answers the purpose of a cushion. So set they have crushed a 6-lb. steel hammer without hurting the jaws. From the crushers the ore goes to the stamp bins. These are so arranged that one bin takes the ore which passes the grizzly, and the next one takes the ore passing the crushers. From here the ore is delivered on to a Challenge feeder. The stamp weighs 750 lb., and drops ninety-six times with a 7-in. to 9-in. drop. The stem is 13 ft. long, and weighs 325 lb. The tappet weighs 100 lb., the boss 210 lb., the shoes 115 lb., and the dies 140 lb. The screens are thirty to the inch of No. 29 steel wire, divided into three sections. The surface of the screen is 14 in. high and 4 ft. 6 in. wide. The stamps drop in the order 2, 4, 3, 5, 1. The cam floor is covered with oilcloth up to the stems to prevent

grease dropping on to the plates below. The shoes wear out first. They are 8 in. in diameter. The height of discharge over the top of the dies is 3½ in. The life of a cast-iron shoe, worn down to 1¾ in., is thirty-five days; of a chrome steel shoe sixty to ninety days, wearing it down 1½ in. to ¾ in. Forged steel wears ninety-five to one hundred and seven days. They have worn as long as one hundred and twenty; they wear flat down to the stem, so that a ring of steel is often broken from the stem of the shoe. Cast-iron shoes wear concave and the dies convex. The chrome steel wears the same way but sometimes trues itself up. The labour in the 60-stamp mill is two men on the crushing floor, which is always kept clean, three men on the battery floor, who work eight-hour shifts and do all the work of the battery, two men on the twenty-four vanners, two men dressing plates, two engineers and two firemen, or thirteen men in all. The mill is greased with what is called the New York compound, one barrel of which, costing $20, lasts for twelve months. The guides, both at the top and bottom of the stamps, are arranged in such a way as to make each one independent, so that a broken stem may be taken out and replaced in three-quarters of an hour without stopping the rest of the battery. The bottom of the top guide has rubber plates to prevent any grease falling into the battery. The stems are scraped every five days to remove any extra grease. All the shafting has 14 in. tin pans underneath the bearings. The stamps use 80,000 gallons of water in twenty-four hours. The whole mill, including the condenser water, uses 170,000 gallons. The mill engine indicates 147 horse-power, and has run ninety days without slowing down. As the mill has sixty stamps there are 1440 battery hours a day, and of these not more than 2½ a month have been lost on account of repairs. No mercury is put in the battery, as it has been found that the material taken from the battery will not amalgamate. The apron of the battery has a plate which has a 6-in. amalgamating surface, and is 2 in. above the battery plate. This plate is of copper, silver electro-plated, with $2 of silver to the square foot of surface. The battery sluice plates are in pairs. They are 8 ft. long, 4 ft. 6 in. wide, and are inclined 9 in. in their length. Their length is determined in every mill by the position at which the almagam ceases to collect. They are dressed every

two hours with a soft whitewash brush, and some little quicksilver sprinkled over them from a bottle with a cloth over the mouth. If the ore contains much manganese, a weak solution of cyanide of potassium is used; otherwise pure water only. To dress the plates, the battery pulp is turned off and clean water turned on. The plates are cleaned once in forty-eight hours, which is done by first softening the amalgam with mercury. The plate is first washed with a stiff brush to take off any dirt. The soft amalgam is rubbed up from below to the centre with a piece of rubber in a flat wooden handle. It is scraped up with a scraper made of a piece of old steel shovel set in a similar handle. When what yields readily has come off, the surface is smoothed over with the rubber; 90 per cent. of the amalgam comes off with the rubber, and 10 per cent. with the scraper. The amalgam runs 980 to 992 fine. The rest is copper. From the battery sluice the pulp falls into an inclined launder, which has riffles at the end. This discharges into a launder 6 in. wide with a copper plate 8 ft. long with riffles every 2 ft. This launder is about 40 ft. long. In the front of the launder is a feather to carry the pulp to the sluice in front, which leads over the second vanner. The vanner supply is from a 1½-in. pipe. Each battery discharges on to two Frues, 13 ft. 4 in. long and 4 ft. 2 in. wide; the vanner belt moves 21 in. a minute, and has an inclination from 2¼ in. to 3 in. in its whole length; the side stroke does not exceed 95 to 98 per minute; 92 is the minimum and 98 the maximum. In front of the concentrator distributor is an apron on which are copper plates, 14 in. to 18 in. long and the full width of the vanner; the separation commences to take place immediately under the sprinkler, and goes on to within 1 in. of the end of the belt. The sprinkler should give a minimum of water. In this mill only every other spout is used, as it was found that the concentration was not so effective when all of them were running. The head should be absolutely uniform, and the speed and supply of water regulated for this purpose; the heads are carried by the belt into a tank filled with water, from which they are washed off, and the overflow from these tanks is settled in two smaller ones; the head tank is emptied at least once in two hours; it is dried and packed in bags; the other two tanks are emptied once a month. The tails

run in launders about 8 in. wide, which are riffled about every 8 ft. to catch-pits. The slimes run off, and the tails are kept in the dams. On these launders are a number of mercury traps, which are cleaned once a month. The tails average 20 per cent. of the value of the ore. The concentrates from the vanners yield from $800 to $1000 to the ton, and are sold to the smelters who charge $13 per ton for treatment, and return the full value of the shipment less 5 per cent. of the value of the silver; the full value of the gold is returned. The amalgam from the batteries and the Frues is collected once a week.

In the main mill the vanners are set on the lowest level; but in the 50-stamp mill, where there is not room enough, the pulp from the battery is elevated to the vanners, which are set on the highest level behind the battery.

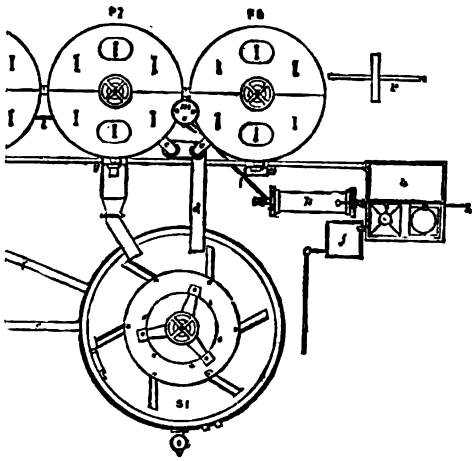
In making a series of experiments on a battery sample from this ore, it was found that 10 per cent. of it remained on a forty screen, 15 per cent. on a sixty, 35 per cent. on an eighty, and 25 per cent. on 150, and 15 per cent. passed a 200 screen.

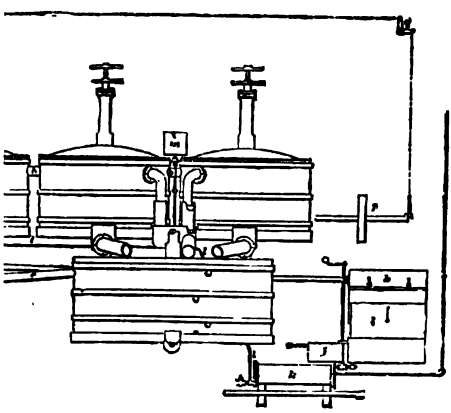
To clean and dress the plates the pulp is turned off and the stamps stopped, and clean water is allowed to flow over the plates until all the pulp is washed off them. Mercury is then sprinkled over them from a bottle with a cloth over the mouth, to soften the amalgam, and the plate is brushed with a stiff brush resembling a clothes brush. The battery plate, which projects 3 in. above it, is then scraped with a common table knife, one man working on each side. It is then rubbed with a rubber with a wooden handle and the amalgam taken up in an iron scoop. The plate underneath the apron plate is then scraped in the same way. It is then rubbed with a rubber two-thirds of the way down on both sides, two widths of the rubber, or a little over 6 in. It is then rubbed towards the middle from both sides. What is collected is taken up in an iron pan holding water, in which all the tools that are used are kept. It is then rubbed down about 12 in. from the edge in the same way, and from the edge it is rubbed up. It is then sprinkled with mercury and rubbed with a soft whitewash brush, and the pulp turned on. When the amalgam is hard it is scraped up with a piece of a shovel set in a wooden handle, bent so that the motion will not disturb the plate. The amalgam is cleaned in

warm water, 100 per cent. of fresh mercury being added to it. If hard it is rubbed in a mortar; if soft, with the hands; this sets all the sulphurets free, which are put to one side. The water is taken off with a sponge, which is constantly wrung out in a kettle of warm water to save any mercury, and the mercury transferred. The skimmings are put into an iron mortar and worked with a pestle and cleaned again with a sponge. This mercury is added to the rest; the sulphurets are put one side. The amalgam is then squeezed in a cloth over an enamelled pot, and made into a ball, which is put one side until there is enough of it to treat. The tools used are one hard brush, one soft whitewash brush, one iron scoop pan, two steel scrapers, two table knives, and two rubber scrapers. These after use are all carefully washed in a pail containing water, which collects the mercury at the bottom. The iron pan with all the tools in it is then put one side until the next clean-up, which is made at the end of every shift. When the batteries are cleaned a considerable amount of gold is found in nuggets from the size of a pea down, which are all carefully collected. No fine gold of such size is ever seen in the ore. The bullion produced in 1888 contained 65 per cent. of gold and 35 per cent. of silver. The cost of mining was $3.25.; the cost of milling in the 60-stamp mill was $1.14, in the 50-stamp $4.07, and in the 10-stamp mill $5.74.

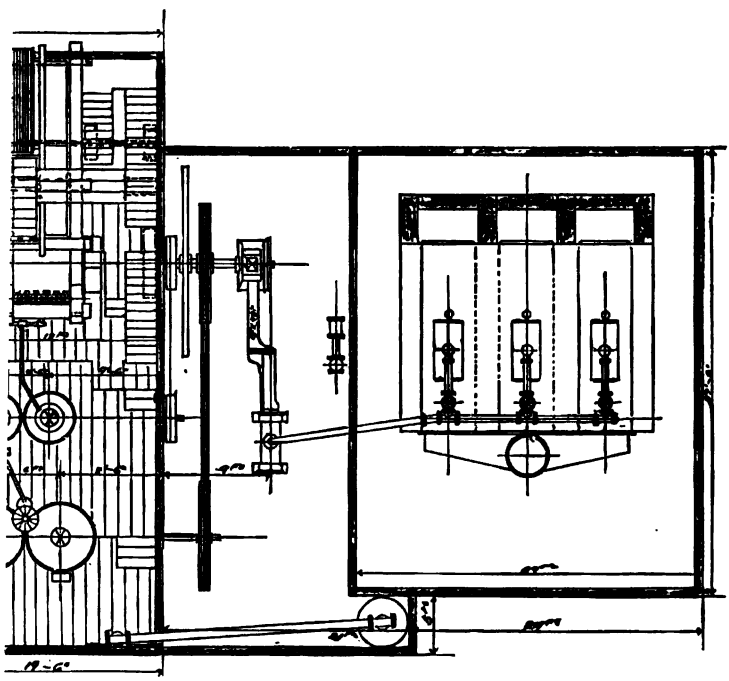
V. The Boss Process.

In the year 1881, M. P. Boss, of San Francisco, patented a continuous process of milling for both gold and silver ores, the object of which was not only to save water but to minimise the amount of labour in the mill, and to save the float gold and the floured mercury, which in the ordinary processes are lost in the tails. For this purpose he causes the ore to be crushed coarse in the stamp mill with the least amount of water requisite to carry the pulp through the screens. The material so crushed is conducted to a series of grinding pans, the mullers of which are always down, in which no mercury is ever charged, and from there through another series of pans in which mercury is introduced, and the mullers of which are always up, and from there through settlers, and from the settlers to concentrating machines if the ore justifies





)LD MILL.



BOSS MILL.

it. By this arrangement of the mill it is made almost automatic. The use of a coarse screen on the battery increases its output, and allows the amount of water used, to be so small as to make the pulp thick enough without settling in the vats, while the pans do the grinding. The pulp is never touched by hand, and as the quicksilver amalgam is often withdrawn, and new mercury is constantly being added, every particle of the pulp is being brought into contact with fresh surfaces of quicksilver, and whatever part both of it and of the gold is carried off from the pans must pass into the settlers, where it is likely to be collected. The number of pans per battery is the same in both systems, but the efficiency is greater when the stream of pulp is continuous. There are two general arrangements of this plan; one, Fig. 72, adapted to old mills where no change is made in the disposition of the works the settlers being left below the pans; and the other, Fig. 73, for an entirely new mill, in this case a 20-stamp mill, where the grinding pans are placed in a line between the stamp and the amalgamating pans, which are also placed in a continuous line with the settlers. It will be seen at once that the latter disposition is the most advantageous, as it has but one line of shafting, and less power is used to do the same work. This system requires less room in the building and less expense in construction, as there will be no necessity for a considerable number of levels in the mill, and consequently almost no grading, and is therefore cheaper to build than an ordinary mill. The pans are generally run by a friction clutch, which avoids a considerable quantity of wear, and any pan can be cut out from the system, the pulp being made to pass from one pan to another by an ejector when for any reason it is desirable to stop it. The process is very economical in labour, as most of it is automatic. There is no labour for charging the pan, and no time is lost in charging and discharging it. The pulp is much more uniform than when settled in vats. The chemicals, as they are regularly introduced, act better and less of them are used, as their action commences at once and is continuous, instead of only being introduced after the pan has been acting for a certain time. The loss in mercury is less and the yield is often as much as from 3 to

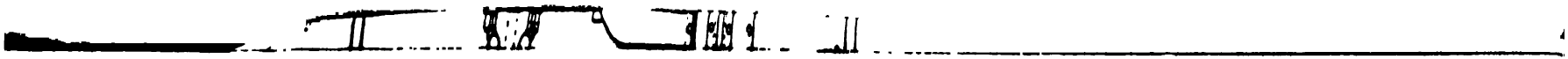
5 per cent. higher than in the old method. As there is no reason for making any dirt, the mill can always be kept clean. As the battery water must pass through the whole system, all the slimes are treated, and the tails can be passed to the concentrators as in any other system. The cost of milling by this process at Minas Prietas in Mexico was $2.12 per ton, while at Calico on silver ores it was $2.60. The yield was 92 to 95 per cent. of the assay value of the ore. As examples of its working in an altered mill the California Mill of Virginia City, Nevada, has been selected; and of its working in a mill constructed specially for it, the Jay Gould Mill at Gould, Montana, has been chosen.

The California Mill at Virginia City is an 80-stamp mill, with the stamps arranged back to back. The stamps weigh 1000 lb., and have ninety-five drops of 7½ in. to 9 in.; the shoe weighs 125 lb., the boss 250 lb., the tappet 125 lb., and the stem 500 lb. The mill is run by a Pelton wheel, 10 ft. 10 in. in diameter, with a 1½-in. nozzle, a head of 605 ft., and a pressure of 263 lb. to the square inch. Tullock's feeders are used; the screens are Nos. 2 and 3 diagonally slotted; each stamp crushes 4 tons in twenty-four hours. From the battery the pulp is run into twelve Boss grinding pans, 4 ft. in diameter, making seventy-five revolutions a minute. From there it runs through a 4-in. pipe 1105 ft. long, to the mill, where there are forty pans, arranged twenty on each side of the building. The mill was built to run on the old system, but was subsequently altered to the Boss system. The pans are 5 ft. in diameter, and make seventy-five revolutions a minute. The power to run the pans is transmitted by wire rope. Four of the pans work with the mullers constantly down; the six last with them always up; 2 lb. to 4 lb. of mercury are added in the first four; in the other six, 200 lb. every two hours; the chemicals are added in the second. The clean-up is made once a month. All the pipes run into the pans on the line of the normal. Sometimes a little lye is added in the pans to cut the grease, from 1 lb. to 10 lb. in twenty-four hours being used. The pans are worked in sets of ten, each set having its own five settlers. On one side they are working the Ray electric process, but get no special results from it, as it works no better than the other side, which does not have it. Of the pans the first

five grind; the others have the mullers up. The mercury is added in the second, and also the chemicals from triangular trays set above the pans, of which there are twelve on a wheel; this wheel revolves once in an hour, tipping a pan every five minutes. In these trays a small dipper of salt is put, and above it a small amount of copper sulphate. The shoes and dies of the first four pans last eighteen to twenty days; those of the others, as they are raised, last five to six months. This process uses the same amount of water as was used in the ordinary process. The pulp is run just as thick as it can be worked. There is, however, a great saving both of labour and of power. The grinding pans which are used in the commencement do their work better and with less power. There is also a saving in room, as the grinding pans are small. The power used for the thirty pans would not run twenty on the old system. The yield of the ore is about 12 per cent. higher than it formerly was. The mill works up from 87½ to 90 per cent. of the assay of the ore. The alteration of the mill from the old to the Boss process took only from twenty to thirty days. From the settlers the pulp goes to the tail sluices. All the mercury in the mill is raised by hydraulic pressure. The loss in mercury in 1880 on $30 ore by the old system was 1.68 lb. In 1847 on $80 to $100 ore it was 2.67 lb.; in 1888 by the Boss system on $30 ore it was 2.25 lb. The pans are run by three Pelton wheels 10 ft. 5 in. in diameter, with a 2-in. and 8-in. nozzle, with a head of 493 ft. and a pressure of 260 lb. to the square inch. Occasionally the indicator shows as high as 600 ft. pressure. The tails from the mill are caught on a series of sluices extending down the cañon. There are six sluices 22 in. wide, and at the end of each sluice they have a drop to a head-box and a box for catching the sulphurets. The first sluice of the set is 300 ft. long, the second is 350 ft., the third 350 ft., and the fourth 400 ft., or 1400 ft. in all. They all have the same grade, and are covered with gunny sacking. They are constantly cleaned from one end to the other, and as soon as the six are cleaned the work is commenced over again, so that all the sluices are cleaned six times in the twenty-four hours. About 40 tons are cleaned up in this way every twenty-four hours when the mill is running 250 tons a day. The material is ground in four pans which make seventy-

five revolutions a minute; 250 lb. of mercury are used for a charge of 2700 lb. It is ground for two hours with the mullers down, when the mercury is added. It is then run three hours with the mullers up. It runs into the settlers for two hours, as there is but one settler for two pans, and what runs over goes to the sluices.

The Jay Gould Mill at Gould, Montana, Fig. 74, which was built as a Boss mill, has ten stamps running on a close-grained quartz, mixed with both an indurated and a soft clay. The quartz contains besides, some pyrites and a little manganese. The proportion of gold to silver is about one to six; the capacity of each stamp is 3 tons. The stamps weigh 800 lb., the shoes 130 lb., the Boss 200 lb., the tappet 125 lb. They drop ninety times 5 in. to 6 in. The shoes and dies are of chrome and wrought steel. No. 4 steel slot screens having a clear cut with about twenty-four to the inch are used. The steel shoes and dies wear eighty to ninety days. The screens last two weeks. The height of discharge is 2 in. No excess of water is allowed to flow into the battery, but only just enough to cause the pulp to pass the screens; the amount of water used being about 60 per cent. of the weight of the ore. All the water used in the mill is heated in the same appliance which heats the water for the boiler. The front of the mortar is closed with a wooden rack, which can be easily taken down. The top is covered with a curtain. The contents of each battery is discharged into a $2\frac{1}{2}$-in. iron pipe, which at the centre between the two batteries joins a $2\frac{1}{2}$-in. pipe, which discharges the whole contents of the stamp into a Boss grinding pan. This is 3 ft. 6 in. in diameter. The shoes and dies are solid with grooves like those of an ordinary millstone cast in them. The muller is set as close to the dies as it can be put and let the ore pass. The bottom of this pan is on a level with the top of the amalgamating pans, of which there are six Boss standard pans set in a straight line. They are run from below with friction gear. These pans have iron bottoms and wooden sides. They are 5 ft. in diameter and 30 in. deep The first two of the six pans are used for grinding with the muller down. To these pans 10 lb. of mercury are added every twenty-four hours, a similar amount having been previously taken out from the mercury received at the side of the



pan at the bottom. The chemicals used are lime and salt, which are all added automatically in the first pan from an automatic machine, on which there are twelve scoops. This is run by a belt from a shaft below, and is regulated so as to tip one of the scoops of chemicals at fixed intervals, the adjustment being made by holes in a lever or crank, so as to make it go fast or slow. The lime and salt are added according to the judgment of the amalgamator. The machine usually makes one revolution in forty minutes, tipping all the twelve pans one by one in that time. The chemicals are added according to the condition of the mercury. An assay is taken in a dipper and carefully washed. The mercury collected is always bright. The salt added is 200 lb. in twenty-four hours, or 8 lb. to 10 lb. to the ton of ore. The pans have iron covers, which are corked down, but holes with movable covers are left to introduce the chemicals and to take the assays. These pans are heated on the bottom with the exhaust steam. The overflow from each pan flows by a 4-in. hole through a conduit to the next one, in which there is the same quantity of mercury as in the first. The next four pans are used for amalgamation. The mullers in these first pans are set so as to just clear the dies. In the others they are raised a ¼ in., so as not to grind the mercury. From these pans, 3, 4, 5, and 6, the mercury is drawn from the quicksilver bowl attached to the side of the pan at the bottom every hour as low as possible without letting out the pulp Then 120 lb. to 150 lb. of fresh mercury is charged from the mercury tank. The pans revolve sixty-eight times a minute. The pulp from pan 6 flows into a settler, which makes sixteen revolutions a minute, into which a constant stream of clear water to dilute the pulp is running from a 2-in. pipe, equal to a ¾-in. pipe with 50 ft. pressure. When the pan foams a stream of water is directed against the muller to break up the foam. The overflow from the third settler goes to the Frues. The mercury in the first settler is drawn every twenty-four hours, and with the mercury from 3, 4, 5, and 6 pans, goes to the strainers, which are automatic. For the clean-up the covers are removed from all the pans. No. 1 is siphoned by a siphon which fits into a mercury receiver and into which a jet of steam is sent from below into No. 2. No. 1 is cleaned. No. 2 is then treated in the same way,

and so on. No. 1 settler is siphoned into No. 2, and Nos. 2 and 3 are siphoned into a discharge tank. The contents of this tank after the clean-up are put into No. 1. The mercury for the pans is lifted by a chain bucket from the strainer into a receiver. The buckets are made of wrought iron attached to a chain, and so constructed that as each one comes to the top the contents are tipped into a spout which runs into the receiver on the side. This causes no spattering when enough water is used. On the mercury tank there is a circular indicator, which shows by a hand the number of times 200 lb. of mercury there is in the receiver. The mercury chain is run by a belt around the main shaft. The loss in mercury is from a ¼ lb. to ½ lb. to the ton of ore. The only place where amalgam is found in the clean-up, sticking to the pan, is in 1, 2, and 3. Occasionally a little is found in No. 4. At the clean-up in July, 1888, only 3 oz. of mercury were found in 4, 5, and 6. A curious backward movement takes place in the pans owing to the centrifugal force, which occasionally throws the mercury moving in the pulp against the sides into the inlet openings, where it collects, and sometimes a considerable quantity of hard amalgam is found in the inlet spout from the grinding pan where no mercury is used. With this process the screen capacity is not great, but no more water than is used can be introduced into the battery for fear of thinning the pulp, as everything must be put through the pan. This is a real saving, as it forces even the slimes to come into contact with the mercury. The grinding is effectually done, as while the battery screens are only forty to the inch, 96 per cent. of the pulp passes 100. The labour in the mill is one man at the breaker in the daytime, one on each shift at the battery, one at the pan, and one at the vanners.

The mill has seven Frues. The ore is of such a character that the speed of the vanners has to be altered every few hours. To ascertain what the work of the battery was a sample was taken and sifted. There remained on the 40 screen 52 per cent., which assayed $32.87; on the 50 screen 8 per cent., on the 60 screen 18 per cent., on the 80 screen 18 per cent., on the 200 screen 1½ per cent., and past the 200 screen 2½ per cent. The last three samples assayed together $196. The average assay of the

whole was $82.40. Of the gold 79.2 per cent. to 83 per cent. is recovered, and of the silver 32.1 per cent. in the pans, or, total gold and silver 78 per cent. Of the total gold and silver 73 per cent. is recovered in the pans and 5 per cent. on the Frues, which makes a total of 78 per cent. The average fineness of the bullion is 988 ; five men only were employed in the mill on each shift of twelve hours, one man at the battery, one at the pans, one at the Frues, one engineer, and one roastabout.

A comparison of the work of the two processes worked side by side on the Comstock in August, 1887, is given below. The Morgan Mill is worked on the old system :

—	Amount Crushed in Tons.	Yield of Ore in Dollars.	Value of Gold contained in Dollars.	Value of Silver contained in Dollars.	Assay Value of the Battery Sample per Ton in Dollars.	Assay Value of the Bullion per Ton in Dollars.
Morgan Mill .	5125	156,038.58	78,209.59	77,828.99	37.89	30.44
California Mill	2644	71,726.27	21,814.58	49,911.69	29.32	27.12

At the Total Wreck Mill in Arizona when a silver ore containing gold was worked on a comparative trial made in August, 1883, on both systems working side by side, the old system gave a yield of 84.12 per cent., and for the Boss system 86.35 per cent.

As the mills in California generally stop on Sunday, this is the day for repairs, and for making general clean-ups. In Nevada, no stoppage of the mills is made at any time, except when repairs or a clean-up is necessary. Generally, however, a clean-up is made once a week. Sunday is the day usually selected for repairs to the battery ; most of the other amalgamating contrivances have to be cleaned every day, some of them twice a day. To clean up the battery, the screens are removed, and the stems driven out of the boss with a steel key, and hoisted out of the way with a Weston pulley block. The shoe and boss are taken out of the mortar, and the corner die pried out with a bar, which allows all the other dies to be easily removed. Everything is taken out of the mortar, which is then thoroughly washed with water, and overhauled, to pick out any amalgam which may have attached itself to any part

of it. The battery sands remaining in the mortar are washed over mercury riffles to catch any gold or amalgam. The fine sands are separated and caught in the pan, the coarse are put back into the battery. The shoes, dies and bosses are looked over, and the amalgam picked out of them. The shoes and dies which need to be replaced, are set on one side to be inspected with great care. If they show any holes they are broken to extract the amalgam from them, before they are sent to the foundry to be recast. The purchase of old shoes and dies and parts of pans and other amalgamating machinery is often a source of considerable profit to the foundries, who not unfrequently find, in breaking them up previous to melting, considerable quantities of amalgam which have escaped detection in the mill. In gold mills, as a very small amount of amalgam is valuable, the least indication of a hole is probed and bored out, or the piece broken, as the amalgam will be likely to collect wherever there are cavities. When all parts of the mortar and amalgamating plates have been thoroughly cleaned the different parts are put together again.

Amalgam from the plates is generally clean enough to simply moisten with mercury, and knead up into 50-oz. balls; but if much iron and sulphurets are present it should be treated as described, which is also the treatment for amalgam from the mortar cleanings and plate skinnings. After washing in small lots in a gold pan and freeing from gravel and large pieces of iron, it is placed in a large wedge-wood mortar, an excess of mercury added, and stirred with the fingers until the whole mass becomes fluid.

Accumulated skimmings, to the fourth part of a retortful, may be put in a retort and distilled slowly; the resulting black-looking product will be received and reduced at the mint or assay office, or can be reduced in a crucible with bi-carbonate of soda as a flux.

The value of the amalgam depends upon the place from which it is taken. That from the mortar is always the richest, its value depending on the state of division of the gold in the ore, being richer with coarse than with fine gold, and varying a great deal in different clean-ups. In Grass Valley the battery amalgam is

worth from $7 to $8.50 the ounce, while the plate amalgam varies from $3 to $4.50.

The cleaned amalgam is placed in a cast-iron retort 12 in. in diameter and 3 ft. to 4 ft. long, this is built into a brick furnace with iron doors, the fireplace of which is 1 ft. by 2 ft. The head of the retort is 24 in. in diameter, and is turned out to receive a cover which is fitted so as to be luted mercury tight, and is

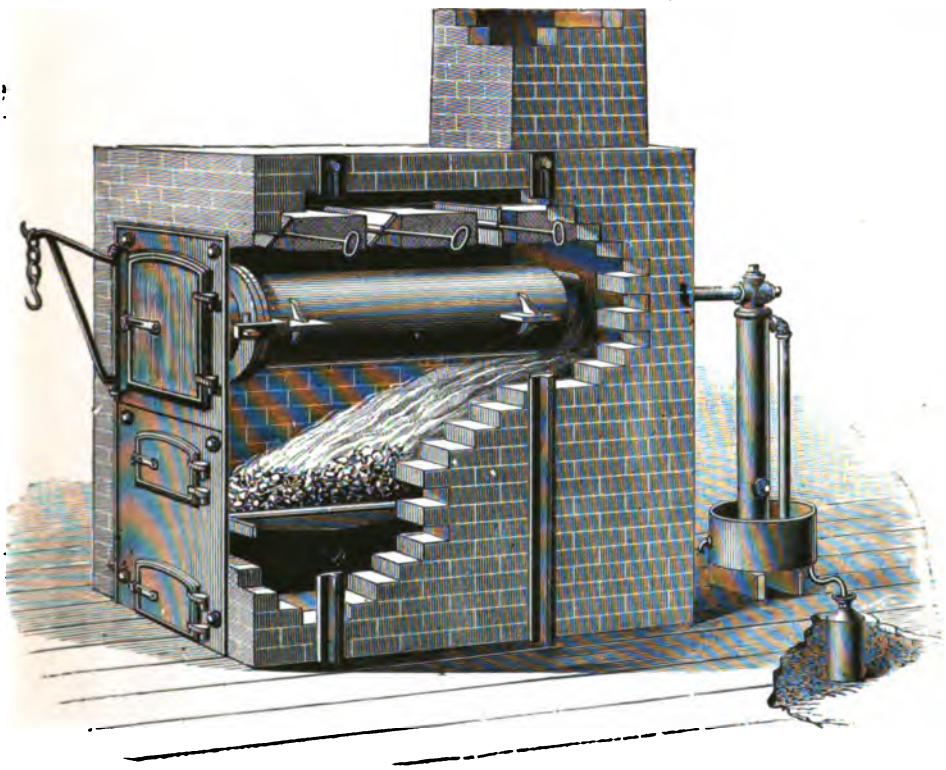


FIG. 75. RETORT FURNACE.

fastened in with a bar or with wedges. The heat passes under and then over the retort, being carried as far to the front of the retort, by means of three dampers, as may be advisable. The chimney is sometimes in the rear and sometimes in the front.

This furnace, Fig. 75, is an improvement on those formerly constructed. It has three dampers for directing the flames and heat from back to the front end of the retort. As the entire retort and its cover is within the furnace the mercury is equally

driven off from all parts of the amalgam, leaving the whole mass equally well retorted. Formerly the front end of the retort was sometimes left entirely empty, and even then the two first trays were often but half done. A crane on the front plate of the charging door is used for lifting out the cover, which enables the operator to open the retort without waiting for it to cool. Four sizes of the retort are made whose weights are given below; the smaller numbers are adapted for gold amalgam, where the capacity required exceeds 1500 oz.

No. 1,	200 lb.	capacity,	all ironwork,	complete,	weighs	1000 lb.
,, 2,	400	,,	,,	,,	,,	1500 ,,
,, 3,	800	,,	,,	,,	,,	2000 ,,
,, 4,	1200	,,	,,	,,	,,	2300 ,,

These retorts are all so made that, when the lower part begins to burn or blister they can be turned round, which makes the life of the retort double that of others that are not adapted for reversing. The condenser delivers its mercury directly into the flask without handling or loss.

The retort usually holds three "boxes" or "cups" made to fit its bottom, each of which has four compartments. These boxes fill the retort. They are placed in the retort charged with amalgam. Sometimes there are no "cups" used. A mixture of salt and ashes is often used to swab out the retort and for luting on the head. At the Jay Gould Mill in Montana the retort when cold is lined with three or four thicknesses of newspaper, and the amalgam is placed on this. The ashes of the paper prevent the adherence of the gold to the retort. When "boxes" are not used the amalgam is placed in the furnace in a large lump divided by discs of iron, which answer the purpose of separating it, so that it need not be broken up afterward when it is to go into the melting pot. The charge is heated slowly to the boiling point of quicksilver and maintained at that temperature for about two hours, until most of the quicksilver has been expelled, when it is gradually heated to a light cherry red for an hour or so, to expel the remainder. It is then cooled down. The time of cooling is from four to six hours. The door of the furnace is then lifted off by the crane or by a pulley, and a pan of sheet iron, 14 in. long and 5 in. deep, with an iron holder 2 ft. long, suspended by four iron rods 5 ft. long, is brought to receive the

gold. This pan exactly fits the exterior of the retort, and is made to rest against its front. The amalgam boxes or the lump "retort" are drawn out upon it by a hook, and are made to swing on to a table covered with sheet iron, 4 ft. long by 3 ft. wide. The amalgam is loosened from the boxes, to which it adheres more or less, by chisels and scrapers made of old files. The scraper is made by turning down the end of a file, and flattening it out, and is 12 in. long, $1\frac{1}{4}$ in. wide, and $\frac{5}{8}$ in. thick. The front edge of the scraper is $1\frac{3}{4}$ in. wide and $1\frac{1}{2}$ in. high. The loosened amalgam is turned out upon the table, and the sides of the boxes are then carefully scraped, and their contents turned out and swept towards the centre of the table with a small hand-broom. The mass is more or less porous, but when the amalgam has been well cleaned it is yellow, though it still contains some silver and sulphides. The retorting of a charge occupies from five to nine hours, according to its size and character. From one-sixth to one-half a cord of wood is consumed per 1000 lb. of amalgam retorted.

When the retorting is properly done the loss of quicksilver by distillation is merely nominal. It will vary from a few ounces to 2.5 lb. to the ton. In the Plumas Eureka Mill, in 1873, it averaged 2.07 lb. At the California Mill about $\frac{1}{2}$ lb. of quicksilver is lost per 1000 lb. of amalgam retorted. Where the vent by which the mercury is carried to the condenser is at the upper edge, or the amalgam is not base, or the pipe is kept open by a rod, the vent does not choke. With dirty or base amalgam there is always danger that the vent may choke if it is in the centre of the end of the retort.

An ordinary retort will stand from 100 to 300 charges if properly handled, but in practice the life of retorts is usually much shorter. Steel retorts have been tried, but it was found that they warped too much. The fact that steel burns out sooner than cast iron would be a sufficient objection to its use.

When all the gold is out, it is weighed and goes to the crucible. The retorted gold at the Keystone Mill from a charge made from a clean-up weighed 67 lb. 10 oz. avoirdupois; the mercury distilled 90 lb. The amalgam weighed 15 lb. 12 oz. The greater part of this loss in mercury was probably in the retort pipes.

About 1 lb. of mercury will probably be collected from that, diminishing the loss of mercury 1 lb. The "retort," as the distilled amalgam is called, will usually weigh from 25 per cent. to 40 per cent. of the amalgam, depending on the state of the gold. It requires much more mercury to amalgamate very fine, than lower grade gold.

The bullion at the Keystone Mill is 840 fine, the impurities being iron, silver, and copper, the most part copper and iron. It will not in California go much below 600 nor much above 850, though cases have been known where it was as high as 920, but such cases are very rare. The gold is melted in crucibles with a flux of carbonate of soda and saltpetre, with a loss of 1 to 1½ per cent. of the assay value, and the name of the mill is then stamped upon it, and it is sold in the market.

In some small gold mills, where the amount of bullion produced is not large, simple pots or bulb retorts of cast iron are used in place of the cylindrical ones. The pots or bulb retorts require no brick fitting, and can be used over an assay furnace or a blacksmith's forge, and are convenient for operations on a small scale.

Both the alembic and cap retorting pots, Figs. 34 and 76, should always be turned out smooth. Before using them, the inside should be well rubbed with chalk. Coating them with moist ashes is bad for the iron, as the alkali will soon honeycomb it. The amalgam balls must be broken up before they are put into the retorting pot, and placed in loosely. The pot must not be more than two-thirds to three-quarters full. Ramming the amalgam hard and solid into the retorting pot, as is sometimes done, is bad practice, as it requires longer to distil out the mercury, and rarely leaves the heart of the retort piece thoroughly distilled.

The flange of the retorting pot may be luted with moistened chalk or sifted ashes, which, when dry, can readily be brushed off. This is applied of the consistency of thick molasses, and the cap slightly turned when put on, to make sure of there being no grit or gravel in the luting. The retort is then keyed up and placed in the furnace, or left on the ground and covered over with fuel. It should be at least 6 in. above the firegrate and be supported by an iron hoop fastened to three legs resting on the grate bars or on the ground

The end of the condensing pipe should never be placed in the water without a cover, as it is apt to result in explosions unless one is an expert at retorting. A canvas or rubber bag should be tied or screwed to its end as shown in Fig. 76 before it is immersed in the water. When the mercury vapour begins

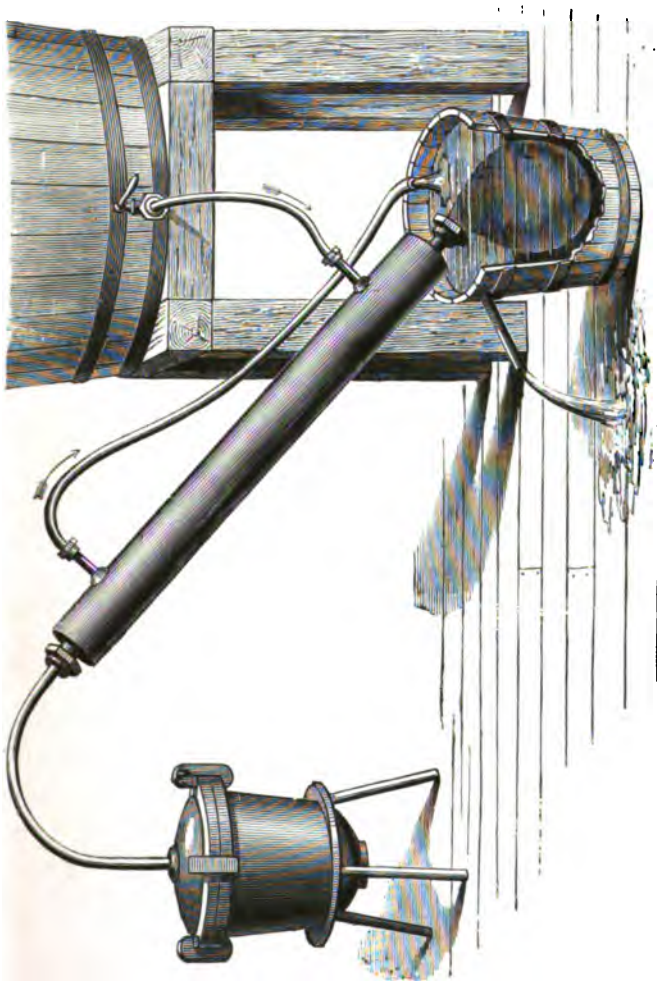


FIG. 76. RETORTING POT AND CONDENSER.

to flow over, the bag will swell up ; this distension is a sign that the mercury continues to come over ; it is continually being condensed, and when the condensation ceases the bag will collapse, indicating the completion of the operation.

The condensing pipe should be kept as cool as possible by

a Liebig's condenser, as shown in Fig. 76, or if that cannot be done by continually pouring cold water over it.

These retorting pots are made in seven sizes, to contain at two-thirds full, from 150 oz. to 2000 oz.

No.	Inside Diameter at Top.	Inside Diameter at Bottom.	Inside Depth.	Capacity in Ounces Two-Thirds Full.
	in.	in.	in.	oz.
1	8	6	12	2000
2	7	6	$9\frac{5}{8}$	1500
3	7	$6\frac{1}{8}$	$8\frac{3}{8}$	1000
4	$6\frac{1}{2}$	5	$8\frac{1}{2}$	800
5	$5\frac{1}{2}$	$4\frac{1}{2}$	7	500
6	$4\frac{7}{8}$	$3\frac{7}{8}$	$5\frac{5}{8}$	250
7	$3\frac{7}{8}$	$3\frac{1}{8}$	$4\frac{1}{4}$	150

Bark makes the best fire for retorting. The fire should be lit at the top and allowed to work down; when it reaches the grate more bark should be put on and the retort kept covered. It should be kept at a bright red until the bag begins to collapse. The fire is then removed and left until it is burnt out and the retort cooled down to blackness.

After twenty or thirty minutes the bag is taken off, the mercury removed and disposed of. The "retort" is lifted out but not opened for at least twenty minutes more. When opened it is turned over on a gold pan or other receiver; and, if the retort gold does not readily fall out, a few light taps on the outside will dislodge it.

This retort metal is easily cut up with a hammer and chisel into pieces of about an ounce weight, and melted in a crucible without flux, as the bar will thereby be cleaner and finer; the slag can afterwards be melted with carbonate of soda, and all the gold saved.

Ingot moulds of fixed capacity, Fig. 77, for any size, for gold bars, are made with a slight taper so that the ingot will readily fall out, and are finished smooth inside with or without round corners. They are generally lettered in bottom with the initial letters of the name of the company, or its full name if it is short.

Adjustable moulds for any shape bar desired are also used, which can be set to hold from 1 oz. to 50 oz. by means of blocks of iron of any desired thickness set between the movable sides.

Cases of sickness through salivation rarely occur. The bursting or imperfect luting of retorts and the cleaning up of pans while still hot are almost the only causes. They were formerly much more common when men who chewed tobacco were allowed to handle the mercury. Now when the mercury is no longer handled they are very rare.

All of the mills have laboratories in which assays of all the samples taken are made. Samples of ore are sometimes taken from each carload at the mine. This is the case at the California

FIG. 77. GOLD MOULD.

Mill, where a handful is taken from each car. When the ore of a mine is treated by a custom mill ore samples are often taken by the mine owners as a check upon the working of the mill.

To know just what is going on in the mills, samples are taken every hour. One from the automatic feeder before the ore goes

FIG. 78. ADJUSTABLE GOLD MOULD.

under the stamps. This contains everything the ore does. One from the battery apron sluices after the ore has passed all the amalgamated plates. These two give all that the mercury can collect. A third is taken from the material passing off as tails from the concentrators. These are all taken wet, the water being evaporated instead of settled, so as to catch all of the slimes. Battery samples are always taken at the mill, and in wet-crushing generally as follows: Every half-hour or hour a rectangular iron box, or dipper, is passed along under the

lips of the mortar where the pulp falls into the sluices, and a portion of the slimes and sand is taken, care being observed to prevent concentration by the overflowing of the box. This sample is put in a bucket or pan with others, and at the end of twenty-four hours the clear water is poured off, or it is evaporated at once, which is the better practice, and the collective samples for the day are prepared for assay.

In dry crushing mills it is usual to take the sample by passing a box along the lower part of the screen every hour or so. Sometimes the sample is taken from the conveyor or from the pulp in the bins.

Tailing samples are taken from one or all of the discharge plugs of the settler or are dipped from the top before each plug is drawn. A very good way is to dip a small gas pipe into the settler close to the top, one end of the pipe being closed with the finger, and withdraw a portion of the pulp.

Silver bullion is sampled by taking a dip from the melted mass after stirring, or a little is poured into water while the crucible is still nearly full, and a little more when it is nearly empty. Either method is good. Gold bullion is sampled by chipping two diagonally opposite corners of the bar after it is cast.

Ore assays are generally made in crucibles, though with very base ores scorifiers are sometimes used. They are seldom corrected for loss in assaying.

To produce $1 in gold costs 8 cents for mill labour in Dakota. The power which is required is 2 to 3 horse-power per stamp. The fuel used is one-tenth to one-fifth of a cord per stamp. The quantity of gold extracted in these mills does not often exceed 80 per cent., and is often much below it. The bullion runs from 600 to 850 fine, the remainder being usually silver. It is not possible without knowing all the conditions to state what yield of ore will pay for working. An ore that could not be treated at all in a small way can often be treated with profit on a large scale. This depends not only on the method of working and on the quantity worked, but also on the character of the ore, which determines the output of the stamp, which may vary from 1½ tons up to, in rare cases, in soft decomposed rock, 10 tons in twenty-four hours per stamp.

It is not always possible to get the commercial details of mining in California. Most of the mines are owned and managed by individuals, local companies, or close corporations, who are not willing to give to the public the details of their business. These details can only be had from stock companies whose reports are published. The cost of milling gold ores in California varies from 54 cents to $2.50 per ton. Where cheap water-power is to be had, ores can be milled at a cost of not more than $1; and where steam-power is used, exceptional facilities may reduce the expenses of working to those of the average water-power mills. In other States and Territories it is as a general thing somewhat more expensive to reduce gold ores; but in the Black Hills region of Dakota the large gold mills are operated so as entirely to overcome the disadvantages of high freights and other inconveniences, because of the simplicity of the process, the scale upon which operations are carried on, and the economy in labour which is effected by the use of automatic labour-saving devices.

Gold-bearing* quartz is now being profitably worked in California, yielding not over $5 per ton; of this grade of quartz there are great quantities in California eligibly situated for treatment. In one instance as low as $0.60 is worked with profit.† Modern improvement and appliances have materially reduced the expenses of extracting and reducing gold ores. With a large number of stamps, water-power, giant powder, powder-drills, rock-breakers, self-feeders, concentrators, and the use of gravity in handling ores, low-grade ores can now be profitably reduced.

The cost of mining varies in different districts according to the condition of the mines, width of ledges, character of country, rock, &c. No two districts are precisely similar, but the rate of labour does not vary materially in California. The operations of a representative low-grade mine have been selected to show the relative cost of different classes of work. The country rock is granite, the vein matter easily broken, the ledge of fair width, and the rates of labour $3 per day. Under these conditions the relative cost was as follows:

* Production of Precious Metals in United States, 1883, p. 723.
† Page 420.

STATEMENT OF COSTS.

Items.	Cost.	Cost per Foot.	Cost per Ton.	No. of Feet.	No. of Tons.	Tons per Foot.	Feet per Ton.
	$	$	$				
Tunnel 7½ by 10, cross-cuts	17,419.22	19.77	3.31	881.00	5,264	5.97	0.167
Levels 8 by 8, opened through tunnel	30,472.88	13.61	2.80	22.38	10,915	4.88	0.205
Winze 5 by 7½	1,254.52	17.92	6.15	70.00	204	2.91	0.343
,, levels 8 by 6	347.68	14.48	3.86	24.00	90	3.75	0.267
Raises 5 by 6½	6,213.90	7.40	2.95	8.39	2,112	2.52	0.397
Cross-cuts 6 by 4½ in vein matter	4,524.28	5.26	2.68	8.60	1,690	1.96	0.509
Stopes	9,588.36	...	2.36	...	4,053		

RATIO OF COSTS.

These items are assumed to cost 100.

Items.	Tunnel.		Levels.		Winze.		Winze Levels.		Raises.		Cross-Cuts.		Stopes
	ft.	tns.	ft.	tns.	ft.	tns.	ft.	tns.	ft.	tns.	ft.	tns.	tns.
Tunnel	100	100	69	85	91	106	73	117	37	89	27	81	71
Levels	146	118	100	100	132	220	107	138	54	105	37	95	84
Winze	110	54	76	46	100	100	81	63	41	48	29	44	38
,, levels	137	86	94	72	124	159	100	100	51	76	36	69	61
Raises	267	112	184	95	242	209	196	131	100	100	71	91	80
Cross-cuts	376	124	259	105	341	229	257	144	141	110	100	100	88
Stopes	...	140	...	119	...	261	...	164	...	125	...	114	100

There is really no standard line of demarcation between the classes of gold-bearing quartz known as "high grade" and "low grade." About 1870 all ores producing by the milling process less than $10 per ton were classed as low grade, and no ore producing less than $20 per ton was regarded as "high grade." Now, ores producing $10 per ton are considered as high grade where water can be utilised as the motive power; and many ledges are yielding a handsome profit where formerly great losses were inevitable. This is due solely to the general adoption of labour-saving machinery and the introduction of high-pressure water wheels instead of steam. The relative cost of the two systems will appear in the returns of several leading companies from various portions of the State. In California, particularly on the "mother lode," the vein in places expands from

TABLE SHOWING OPERATIONS OF SOME OF THE LEADING GOLD QUARTZ MINES IN CALIFORNIA DURING THE YEAR 1883.*

Name of Mine.	Location of Mine.	Average Number of Miners Employed.	Miners' Wages per Day.	Cost of: Sinking per Foot.	Cost of: Drifting per Foot.	Cost of: Mining per Ton, including Dead Work.	Cost of: Milling per Ton.	Depth of Mine in Feet.	Number of Tons Extracted and Worked.	Average Yield per Ton.	Per Cent. of Sulphurets.	Total Bullion Production.	Remarks.
			$	$	$	$	$			$		$	
Patterson .	Tuttletown, Tuolumne Co.	Shaft, 6 Stopes, 8	1st class, 3.00 2nd class, 2.50	15.00	2.50	1.60	1.00	...	7,141	4.38	3	30,561	Two months lost placing pumps in shaft. Short of water for three months and did not run at full speed.
Sumner, Big Blue, Lady Belle, Bull Run, and eight others	Kernville, Kern Co.	65	3.00	Shaft, 75.00 Winze, 12.00	8.00	2.75	1.75	400	12,000	9.00	2½	108,000	Hoisting works 6 engines, 16 in. to 12 in. pumps. Adit tunnel 2000 ft. long, for draining 150 ft. from bottom.
Sierra Buttes	Sierra Co.	180	†65.00	...	...	4.97	.59	...	31,580	6-7	1	...	Hand-feeding to batteries.
Plumas Eureka	Plumas Co.	200	†65.00	...	...	3.55	.45	...	57,194	6-7	1	...	Automatic battery feeders.
Shepley .	Ophir, Placer Co.	8	3.00	12.00	3.50	2.50	2.00	300	2,000	25.00	2		
Belmont .	,, ,,	12	3.00	12.00	4.00	2.00	1.50	150	...	15.00	2		
Gold Blossom	,, ,,	20	3.00	15.00	5.00	3.50	1.50	200	...	15.00	5		
Saint Patrick	,, ,,	30	3.00	20.00	6.00	3.50	1.50†	800	...	22.00	2		

* Prod. Precious Metals in the U.S., 1883, p. 232.

†½ Per month.

TABLE SHOWING OPERATIONS, &c., DURING THE YEAR 1883—*continued.*

Name of Mine.	Location of Mine.	Average Number of Miners Employed.	Miners' Wages per Day.	Cost of				Depth of Mine in Feet.	Number of Tons Extracted and Worked.	Average Yield per Ton.	Per Cent. of Sulphurets.	Total Bullion Production.	Remarks.
				Sinking per Foot.	Drifting per Foot.	Mining per Ton, including Dead Work.	Milling per Ton.						
			$	$	$	$	$			$		$	
Idaho . .	Grass Valley, Nevada Co.	216	3.00	25.00	9.00	9.21	...	1200	28,572	12.76	½	364,599	Water-power introduced in place of steam, saving $25,000 per anm.
Original Empire . .	Grass Valley, Nevada Co.	45	3.00	20.00	6.50	9.00	1.25	1425	...	...	3	...	Shaft at incline of 30 deg.
New York Hill	Grass Valley, Nevada Co.	62	3.00	25.00	10.00	15.00	2.10	1300	4,876	19.00	3	...	Produced 177 tons of concentrates yielding $62 per ton at custom works. Much dead work done.
Consolidated Wyoming .	Nevada District, Nevada Co.	40	3.00	15.00	4.00	3.50	1.50	700	6,800	...	3¼	...	Worked by funnel and shaft.
Murchie .	Nevada District, Nevada Co.	25	3.00	10.00	4.00	5.00	1.50	720	7,900	17.75	4	...	Worked by shaft.
Nevada City	Nevada Co.	33	3.00	7.00	2.00	3.70	1.05	...	...	15.00	3		
Merrifield .	Nevada City, Nevada Co.	45	3.00	...	...	3.00	1.75	900	...	...	5		
Providence .	Nevada City, Nevada Co.	90	3.00	...	2.00	4.00	.92	1700	...	...	7		
Canada Hill, L. Charonnat and Sons .	Canada Hill, Nevada Co.	40	3.00	20.00	3-5	10.00	1.45	...	5,000	20.00	2½–3	...	Water-power introduced in 1884, will reduce milling to $1.

TABLE SHOWING OPERATIONS OF SOME OF THE LEADING QUARTZ MILLS IN CALIFORNIA DURING THE YEAR 1883.

Name of Mill.	Location of Mill.	Power.	Stamps.				Pans.	Concentrators.	Cost of Mill.	Twenty-four Hours' Capacity in Tons.	Cost of Treatment per Ton.	Tons Crushed during Year.	Method of Treating Sulphurets.
			Number.	Weight in Pounds.	Drop in Inches.	Drops per Minute.							
									$		$		
Patterson	Tuttletown, Tuolumne Co.	Water, high pressure.	20	750	4½–5	85	Drakes	Schofield	...	30–40	1.00	7,141	Chlorination.
Sumner	Kernville, Kern Co.	Water, turbine wheel	80	750	7	78	5	16	120,000	120	1.75	12,000	Pans and settlers.
Sierra Buttes	Sierra Co.	Water	50	750	7½	84	...	...	...	90	.59	31,580	
Plumas Eureka	Plumas Co.	,,	60	750	8	80	...	...	...	150	.45	57,194	Pans.
Shepley	Ophir, Placer Co.	,,	10	750	8	80	...	1	16,000	15	1.50	2,000	Sold to ore buyers, value $110 per ton.
Belmont	,, ,,	,,	10	800	7	95	...	4	15,000	20	1.50	...	Sold to ore buyers, value $300 per ton.
Gold Blossom	,, ,,	,,	10	800	8	85	...	4	15,000	20	1.50	...	Sold to ore buyers, value $350 per ton.
St. Patrick	,, ,,	,,	15	750	8	80	2	6	15,000	22	1.50	...	Sold to ore buyers, value $150 per ton.
Idaho	Grass Valley, Nevada Co.	,,	35	850	9½	72	Handy	Cornish	40,000	100	*9.21	28,572	Chlorination.
OriginalEmpire	,, ,,	Steam	20	800	7	85	1	6	...	40	...	...	,,

* Includes mining.

TABLE SHOWING OPERATIONS OF SOME OF THE LEADING QUARTZ MILLS IN CALIFORNIA DURING THE YEAR 1883—*continued.*

Name of Mill.	Location of Mill.	Power.	Stamps.				Pans.	Concentrators.	Cost of Mill.	Twenty-four Hours' Capacity in Tons.	Cost of Treatment per Ton.	Tons Crushed during Year.	Method of Treating Sulphurets.
			Number.	Weight in Pounds.	Drop in Inches.	Drops per Minute.							
Omaha . .	Grass Valley, Nevada Co.	Water and steam	10	800	9	70	1	4	$ 12,000	20	$ 1.50	...	Treated at custom works at $20 per ton.
Merryfield .	Nevada City . . .	Water	30	750	7	90	...	12	40,000	60	1.75	...	Chlorination.
Providence .	,, ,, . . .	,,	40	750	7	92	...	16	56,000	80	.92	...	,,
Nevada City.	,, County . .	,,	20	750	6	93	...	8	50,000	35	1.05	...	,,
Wyoming .	,, ,, . .	,,	10 6	750 900	6½	94	...	8	24,000	26	1.50	...	,,
Murchie .	,, ,, . .	Steam	10	750	6½	90	...	4	15,000	19	1.50	6,480	,,
Canada Hill, L. Charannot and Sons .	Canada Hill, Nevada Co.	,,	10	750	7	100	...	4 Frue vanners	16,000	10	1.45	5,000	,,

50 ft. to 100 ft. in width, or at least it is metal-bearing for that width, inclusive of the slates, giving an average value per ton of from $4 to $6. In many places it can be worked as an open quarry to a considerable depth, and by adit level to a much greater depth. The great saving is in labour, as a large mill can be run at but little more expense than a small one. In a mine of this class situated in Tuolumne County, on the section of the mother lode beginning at Rawhide and running to the bank of Wood's Creek, below Jamestown, for several miles of the lode the rock worked in bulk yields in the mill from $3 to $6 per ton, and everywhere in this section it can be mined at an expense of $1 per ton. At the Spanish Mine, Nevada County, California, the total cost of mining and milling is $0.50, but this is an exceptional case.

The Tables on the preceding pages give the cost of mining and milling at seventeen of the principal quartz mines in California, and the number of tons of ore crushed down to the year 1882 at some of these mines.

The cost of mining and milling ore at these mines was about the same in the year 1886.* At some of them, as, for instance, the Plumas Eureka and the Sierra Butte tunnels have since been driven, through which the mines are drained and the ore extracted, saving as much as a dollar per ton in the cost of bringing it to the surface. But this gain has been offset by the greater depths attained in other mines. Taking this entire group of mines there has probably been a gain effected of 2 per cent., however, in the cost of mining and milling these ores since 1883, principally owing to improved machinery in the mines and methods and processes which have been introduced for working the ore.

A rough division of the gold-bearing quartz mines of California into three classes may be rationally made. First, small veins yielding high grade ore in comparatively small quantities, whose cost of extraction and milling is $25 per ton. The average yield, say, of such mines would be about $50 per ton. As an example, the New River district in Trinity County: The quantity of this class of ore amounts to 5 per cent., or a little more, of all

* Letter from W. A. Skidmore, of San Francisco.

the ore extracted in the State. The second class would embrace all the large mines, including 75 per cent. of all the ore extracted in the State. The average cost of this class would be about $5 per ton to mine and mill. Besides the mines mentioned in the Tables, this would include such mines as the Plymouth and the Zeile on the "mother lode" in Amador County, whose ore assays only $4 to the ton. In the third class, which comprises 20 per cent. of all the ore mined in the State, would be included the Bodie mines, whose grade is rather high, yielding $30 per ton, expensive to mine and mill on account of the high prices of labour and fuel, lumber, &c., the average being about $12 per ton. In this class might perhaps be included the silver mines of the Calico and other districts of San Bernardino County; also most of the gold and silver mines in Inyo County. Labour, fuel, lumber, and mining supplies cost from 20 per cent. to 40 per cent. higher in these districts than in most other parts of the State. The Tredwell Stamp Mill in Alaska, which formerly had 120 and now has 240 stamps, has been running on $8 ore, extracted and milled at a cost of $1.25 per ton. The mill is driven by water-power. Most of the labour is Chinese and Indian. The ledge is 150 ft. wide. The ore is quarried. The mine is perfectly dry. Ships discharge their cargoes within 400 yards of the mine, and the mill is supplied with every manner of labour-saving device. This is an unusual case.

Prior to the year 1865 it was considered essential that vein matter should yield at least $20 a ton to be treated with profit, but the introduction of modern machinery has shown that an ore yielding $8 to $10 can be mined at great depths and profitably milled when steam is used, and when high-pressure water-power can be had $5 is profitable. Miners' wages are usually $3 to $4 per day. Wood for steam varies from $2 to $12 per cord, and water-power from 16 cents to 32 cents per ton of ore treated.

The cost* of mining and milling in some of the mines in California in 1885 was:

* Mining and Scientific Press, vol. lii., p. 61.

——	Number of Stamps.	Quantity Crushed in 1884.	Yield per Ton.	Cost of Mining and Milling per Ton.	Power.
			$	$	
Idaho . .	35	31,143 tons	18.04	7.86	Water.
Delhi . .	8	20 tons a day	14.00	2.50	,,
Alaska . .	20	30 ,, ,,	25.00	6.75	,,
Sierra Butte .	90	30,000 tons a year	6.47	4.30	
Young America .	8	15 tons a day	60.00	4.00	,,
Plumas Eureka .	60	... ...	7.27	4.32	,,
Grand Victory .	50	225 tons a day	2.00	.75	
Plymouth . .	120	80,000 tons a year	13.00	3.89	,,
Zeile . . .	40	120 tons a day	4.50	2.00	,,

In Amador County the average cost is not over $4 a ton.

The cost* of mining for a month at the Boston 20-stamp mill at Mokalumne Hill, Calaveras County, California, in 1879, was:

	$
Water, at $10 per day for twenty-nine days	290
One foreman, per month	100
Two amalgamators, $50 each	100
,, concentrators, $50 ,, ...	100
One rock-breaker	50
,, helper	50
Lights, fuel, charcoal, assays, &c.	150
Wear and tear	300
Total	$1140

or about $1 per ton for crushing, amalgamating, and concentrating.

The following statement of expenses of working a mine with a 10-stamp mill—the whole plant being run by steam-power—is taken from the books of the Murchie Gold Mining Company, for 1882:

Items of Mining Expense—Steam Power.†

	$
Superintendence, including bookkeeping and assaying, per day	12.00
Mine foreman (for day time), per day	4.00
,, ,, (for night time)	4.00
Blacksmith	3.50
,, helper	1.50
Engineer	3.25
Lander	2.50
Fireman	2.00

* Prod. Gold and Silver in the U.S., 1880, p. 339.

† Prod. Precious Metals in the U.S., 1883, p. 728.

Mill expenses:	$
Amalgamator, per day	4.00
Two millmen	5.00
One Chinaman	1.50
Screens	0.80
Quicksilver	0.40
Fuel, per cord	3.50
Powder, per pound	0.23
Candles, per pound	0.18
Steel, per pound	0.14
Iron, per pound	0 4-5
Fuze, per 100 ft.	0.67½
Mine timbers (foot)	0.05
Lagging, 6 ft. each	0.05
Poles, 6 in., 20 ft.	1.00
Oils, gallon	1.00
Charcoal, bushel	0.12
Steel-wire rope (foot)	0.50

The cost of fuel and the necessary labour attending the use of steam-power at the above mine is $24.50 per day, the consumption of fuel averaging 200 cords per month. The segregated cost of milling is $1.50 per ton. But with water-power a saving of 75 cents per ton could be effected on a 40-stamp mill. The record would thus stand approximately:

	$
Mining, per ton	3.50
Milling ,,	0.75
Chlorination of ore, per ton (rock broken from the mine)	0.35
Estimate dead work to ton of ore	0.40
Total	$5.00

The capacity of a 40-stamp mill may be estimated at 28,000 tons per annum.

In Nova Scotia* the cost of mining a 12-in. quartz seam with walls 3 ft. apart is $4.95; with a 6-in. seam it would be a little more than double, and on a 2-ft. seam a little less than half the cost of a 12-in. The total cost for mining and milling would be:

	$	
Mining	4.95	
Hoisting	0 57	
		5.52
Crushing	0.76	
Fuel	0.32	
Hauling to mill	0.05	
Supplies, wear and tear	0.15	
		1 28
Total cost of mining and milling		$6.80

* Trans. Am. Inst. Min. Eng., vol. xiii., p. 663.

The following Table gives the labour and cost of mining and milling in some of the mills of California and elsewhere, and shows how the labour account diminishes with the size of the mill:

Name of the Mine.	No. of Stamps.	No. of Men in the Mill.	Cost of Mining.	Cost of Milling.
			$	$
North Banner .	5	2	...	0.75
Suffolk . . .	5	2	...	1.10
Moore . . .	10	4	1.75	0.80
Sutter Creek .	10	2	0.94	1.11
Stickels . .	20	3	2.05	0.40
Buchanan . .	20	6	3.00	1.25
Washington . .	20	4	2.50	0.40
South Spring Hill	30	5	2.50	0.65
Sheep Ranch .	30	7	3.00	1.00
North Star . .	30	6	5.00	
Idaho . . .	35	18	5.25	2.40
Bunker Hill . .	40	5	3.35	0.60
Keystone . .	40	9	4.00	0.75
Kenedy . .	40	6	3.00	0.69
Young America .	40	10	2.00	0.75
Utica . . .	60	16	1.50	0.85
Plumas Eureka .	60	12	3.41	0.48
Treadwell 120 .	120	28	0.75	
Plymouth Consolidated . .	160	...	2 34	0.39
Treadwell 240 .	240	40	0.75	0.50

The following Tables* give the weights of the different parts of mills. The weights of the ore-feeders and the concentrators depend upon the pattern used, but these figures give a basis upon which to estimate freights.

Weights of Five-stamp Gold Quartz Mill.

	lb.
Rock-breaker	4500
Ore-feeder ("Challenge")	750
Mortar, for 750-lb. stamps	4000
Dies, 100 lb. each, five	500
Shoes, 110 lb. each, five	550
Stems, 340 lb. each, five	1700
Bossheads, 200 lb. each, five	1000
Tappets, 100 lb. each, five	500
Cams, 150 lb. each, five	750
Cam-shaft	350
Carried forward	...14,600

* Joshua Hendy's Machine Works.

	lb.
Brought forward	14,600
Concentrators ("Triumph"), 2275 lb. each, two ...	4,550
Clean-up pan	800
Revolving barrel	1,200
Batea	200
Retort	75
For pulleys, piping, belting, bolts, &c.	2,400
Total	23,825

Weights of a Ten-stamp Gold Quartz Mill.

	lb.
Rock-breaker	5,200
Ore-feeder ("Challenge"), 750 lb. each, two	1,500
Mortar for 750-lb. stamps, 4000 lb. each, two	8,000
Dies, 100 lb. each, ten	1,000
Shoes, 110 lb. each, ten	1,100
Stems, 340 lb. each, ten	3,400
Bossheads, 200 lb. each, ten	2,000
Tappets, 100 lb. each, ten	1,000
Cams, 150 lb. each, ten	1,500
Cam-shaft, 13 ft. 6 in. by 4 ft. 1 in.	700
Concentrators ("Triumph"), 2270 lb. each, three ...	6,810
Clean-up pan	800
Revolving barrel	1,200
Batea	200
Retort	100
Pulleys, piping, belting, bolts, &c....	5,300
Total	39,810

Weights of a Twenty-stamp Gold Quartz Mill.

	lb.
Rock-breaker	10,300
Ore-feeder ("Challenge"), 750 lb. each, four	3,000
Mortar for 800 lb. stamps, 4,500 lb. each, four... ...	18,000
Dies for 800 lb. stamps, 125 lb. each, twenty	2,500
Shoes for 800-lb. stamps, 120 lb. each, twenty... ...	2,400
Stems for 800-lb. stamps, 360 lb. each, twenty ...	7,200
Bossheads for 800-lb. stamps, 210 lb. each, twenty ...	4,200
Tappets for 800-lb. stamps, 110 lb. each, twenty ...	2,200
Cams for 800-lb. stamps, 165 lb. each, twenty... ...	3,300
Cam-shaft	1,700
Clean-up pan	1,000
Revolving barrel	1,400
Batea...	200
Retort	300
For pulleys, belting, piping, bolts, &c.	12,600
Total...	70,300

Weights of Machinery and Appliances Used in the Construction of Quartz Mills.

	lb.
Grizzly	700
Rock-breaker, "Challenge," No. 2	4,500
,, ,, No. 3	5,200
,, ,, No. 4	10,300
Ore-feeder, "Challenge"	750
,, ,, "Stanford"	600
Mortar, for 750-lb. stamps (of size for five stamps) ...	4,800
Dies, for 750-lb. stamps	100
Shoes, for 750-lb. stamps	110
Stems, for 750-lb. stamps	340
Bossheads, for 750-lb. stamps	200
Tappets, for 750-lb. stamps	100
Cams, for 750-lb. stamps	150
Cam-shafts, for 10-stamp battery, 13½ ft. long by 4½ in. diameter	700
"Hendy" Concentrators	1,100
"Triumph" Concentrators...	2,270
Grinding pans, 4 ft. to 5 ft. diameter ...	1,800 to 2,500
Settlers, 6 ft. to 8 ft. diameter	1,600 to 2,400
Clean-up pans	600 to 1,000
Revolving barrels	1,200 to 1,800
Bateas, iron	200
Retorts, 200 lb. to 200 lb. capacity ...	1,000 to 2,400

The following Table gives the material required for both silver and gold mills :*

Ten-stamp Free Ore Silver Mill.

One 4 ft. by 12 ft. grizzly.
,, 8 by 10 Blake crusher.
Two automatic ore-feeders.
One 10-stamp battery, 750 lb. to 800 lb. stamps.
Four 5-ft. combination amalgamating pans.
Two 8-ft. settlers.
One 3-ft. clean-up pan.
,, amalgam safe and strainer.
,, quicksilver elevator, with tanks, pipes, &c.
Two travelling crabs for battery and pans.
One 14-in. silver retort.
,, melting furnace.
Shafting, pulleys, boxes, &c., for mill.
Belting for mill.
Pipes and fittings complete.
One Duncan concentrator for saving quicksilver and amalgam.
Weight of the above, 87,000 lb.

* 6th Report of the State Mineralogist of California, Part II., p. 89.

Battery pipes and fittings.
Belting for mill.
One gold retort.
Weight of the above, 72,000 lb.

Power required:
One 50 horse-power engine.
,, 50 horse-power boiler.
,, 50 horse-power feed-water heater.
,, No. 3 steam pump.
Steam and water connections.
Weight of the above, 24,500 lb.
Total weight of the above, 96,500 lb.

TWENTY-STAMP GOLD MILL.*

Specifications of Twenty-stamp Gold Quartz Mill for the Buchanan Mine, Tuolumne County, California.

One (1) grizzly, to be made of ¾-in. by 2½ in. wrought iron, 12 ft. long 4 ft. wide; spaced 2 in. between the bars, and secured by three ¾ in. bolts, and cast-iron thimbles and washers; also end bars.

Rock-Breaker.—One Blake crusher, fitted with two flywheels, 4 ft. in diameter, and one (1) pulley, 30 in. in diameter, and 12-in. face crankshaft, to have out-board bearings; holding-down bolts for each, to be long enough for 14-in. timbers; all caps for boxes to be fitted to use zinc oil-cups and "Albany" lubricating compound.

Battery.—Four (4) single discharge gold mortars, each weighing about 5000 lb.; the bottoms to be 6½ in. thick; well planed; the screw frame bearings to be planed also; the bases to be 26 in. wide, 3 in. thick, and cored for eight (8) mortar bolts 1½ in. in diameter; the feed mouth to be 26 in. long and 3½ in. wide at the smallest part; the mortars to be well finished, and to be 56 in. long and about 52 in. high.

Mortar Bolts.—Thirty-two (32) mortar bolts, 1½ in. in diameter and 36 in. long; nuts on both ends, and thirty-two (32) 4-in. square countersunk washers for the lower ends; one (1) wrought-iron wrench; jaw designed for the nuts of mortar bolts; arm to be 6 ft. long, slightly curved, made of 1 in. by 2½ in. iron.

Stamp Dies.—Stamp dies, 7 in. deep, 8½ in. in diameter, with square bases, having bevelled corners, and all made to properly fit the mortars, and to be cast of the best car-wheel iron; stamps to weigh 850 lb. each.

Shoes.—Twenty (20) stamp shoes, 8½ in. in diameter, 7½ in. long; to be cast of the best car-wheel iron.

Heads.—Twenty (20) stamp heads, 8½ in. in diameter, 18 in. long; key-holes to be 1⅛ in. by 3 in.; one (1) key for driving out shoes and stems, and to be made as may be directed.

Tappets.—Twenty (20) double-faced gib tappets, made of the best steel faces, 9 in. in diameter; tappets to be 12 in. long; bands or flanges turned, and each to have gibs and three (3) steel keys, and all to be marked; keys to be boxed for shipment.

* 6th Report of the State Mineralogist of California, Part II., p. 90.

Cams.—Twenty (20) double-armed cams, made of best steel; ten (10) to be right, and ten (10) to be left-handed; hubs on one side, and to be strongly banded with ¾ in. by 2 in. wrought-iron bands, well shrunk on; the faces of the cams to be well smoothed, and to be fitted to 5½-in. cam-shafts, and to be properly marked.

Feeders.—Four (4) Hendy Challenge ore-feeders, complete with all latest improvements; also twenty (20) steel keys for same, to be marked and fitted, and properly boxed for shipment; the cams to be about 32 in. long, with 2½-in. face, and struck to give an 8½-in. drop if required.

Corner Boxes.—Six (6) corner boxes, for 5½-in. cam-shafts; cored for 1-in. bolts; backs to be planed true; and bearings to have a strip running lengthwise, and the balance of the bearings to be well babbitted and bored; the ends to be faced; the cap to be solid, bored, but not babbitted unless required, and bored with three (3) ⅝-in. holes each, for using "Albany" compound.

Cam-shafts.—Two (2) camshafts, 5½ in. in diameter and 14 ft. 6 in. long, and key-seated between bearings.

Jack-shaft.—Four (4) jackshafts to be 3 in. in diameter and 59 in. long, with eight (8) bearings for same, and to be made of cast iron.

Stems.—Twenty (20) stamp stems 3½ in. in diameter, 14 ft. long; turned and tapered off both ends for heads; made of rolled iron.

Latch Sockets.—Twenty (20) open latch sockets; all to be well lined with heavy leather.

Guides.—Four (4) complete sets of best white oak guides. The lower guides to be 16 in. wide and 4 in. thick, and the upper to be 14 in. wide and 4 in. thick; bored for 3½-in. stamp stems, and to be 10 in. between centres; all to be 60 in. long.

Cam-shaft Pulleys.—Two (2) wood pulleys, 72 in. in diameter, with 16-in. faces and 6-in. thickness of wood between the flanges, and to be made of the best kiln-dried sugar pine; to be turned and well bolted to 40 in. in diameter cast-iron sleeves; flanges for 5½-in. camshaft; the flanges to be faced.

All of the keys for this mill to be made of the best steel.

Overhead Crabs.—One (1) crab for battery having flanged wheels 7 in. in diameter and 2-ft. face; the axles to be 1½ in. square and long enough to set the wheels 10 in. between the flanges; the boxes to be 1¼ in. thick, and all made of the best wrought iron.

Piping, &c.—All piping, hose, valves, bibs, cocks, ells, tees, unions, &c., to be furnished as per detailed bill. All necessary bolts, belts, nuts, and washers, also copper plate for battery and aprons, to be furnished of proper weight, and as shown on drawings, ⅛ in. thick, not plated; also, silver-plated copper plate, 20 in. wide and 20 ft. long.

Belting.—One (1) main driving belt, to be about 55 ft. long, 20 in. wide, and made of the best five-ply rubber of the "Boston Belting and Packing Company's" manufacture, patent stretched.

Battery Belts.—One hundred and ninety (190) feet of the best five-ply rubber, 16 in. wide.

Rock-Breaker Belt.—Ninety (90) feet of four-ply rubber, 12 in. wide.

Boxes for Shafting.—All boxes to be planed on the bottom and faced on the ends, and to be well babbitted, and the caps to be bored with $\frac{5}{8}$-in. holes, for Tatum and Bowen's lubricating compound. Two (2) steel collars and set-screws for each.

Shafting and Pulleys.—One (1) main battery line shaft to be made in two lengths, 4 in. in diameter, 20 ft. long, and $3\frac{1}{2}$ in. in diameter, 20 ft. long, coupled together with solid coupling. One (1) countershaft for driving rock-breaker, 3 in. in diameter, 20 ft. long; both shafts fitted with all necessary bearing boxes, and holding down bolts for 14-in. timbers.

Pulleys.—One (1) main driving pulley to be fitted to engine crankshaft at mine, to be 48 in. in diameter, 21-in. face.

One (1) pulley for main battery line shaft, to be 72 in. in diameter, 21-in. face, 4-in. gauge.

One (1) pulley to drive crusher countershaft, 40 in. in diameter, $12\frac{1}{2}$-in. face, 4-in. gauge.

One (1) pulley to drive battery, 34 in. in diameter, $16\frac{1}{2}$-in. face, $3\frac{1}{2}$-in. gauge.

One (1) pulley to drive crusher, about 40 in. in diameter, $12\frac{1}{2}$-in. face, 3-in. gauge.

One (1) pulley on crusher countershaft, 26 in. in diameter, $12\frac{1}{2}$-in. face, 3-in. gauge.

Tighteners.—Two (2) battery tighteners, 16 in. in diameter, 20-in. face, with racks, pin-irons, frames, hand-wheels, pawls, and pawl-plates, complete for stopping and starting the batteries.

One (1) rock-breaker tightener complete, as above specified, for stopping or starting the crusher.

All necessary bolts for batteries and building, and all necessary belt rivets for securing the belts. All belting used in this mill to be of Boston Belting Company's manufacture, best quality and patent stretched.

All castings to be made of good material, smooth and free from blow-holes or other imperfections. All keys to be made to fit and properly marked.

Mortars are to be cast from selected patterns or drawings, which will be furnished.

Should anything be lacking to complete this mill which has not been specified, it must be supplied the same as if fully mentioned.

Everything to be delivered on the ground (at company's expense), and timbers to be cut according to plans to be supplied.

Note.—The company already had an engine and boiler. The building, also, over the mill was built by the company. The cost of the above material was $5985, and of the labour, $2300. Save a slight change in the tappets, the work was satisfactorily done, and the mill crushed over two tons per stamp of hard quartz every twenty-four hours.

The total cost of grading, building, machinery, &c., was a little short of $15,000.

GENERAL SPECIFICATION OF A TEN-STAMP BATTERY.*

2 High cast-iron mortars, of latest improved pattern, single (or double)

* Fraser and Chalmers, Chicago.

discharge (plates 20 or 21), planed on bottom, drilled by template, and seats for screen frames planed.
2 Screens, of hard wood, fitted to mortars.
4 Wrought-iron keys for same.
2 Screens of Russia iron or wire cloth.
2 Sheets of rubber for mortar foundations, ¼ in. thick.
10 Stamp dies.
10 Stamp shoes.
10 Stamp heads or bosses, bored for stems.
10 Stamp stems of refined iron, both ends tapered and fitted to heads.
10 Stamp tappets, fitted with wrought-iron gib and two steel keys in each.
10 Cams, right and left hand, double lift, hubs banded and fitted to shaft with steel keys marked to place.
1 Cam-shaft of hammered wrought iron, turned full length, key-seated and marked where cams are fitted.
3 Cam-shaft boxes, babbitted, bored, and planed on bottom and back.
2 Cam-shaft collars of wrought iron, with steel set-screws.
1 Cam-shaft pulley, built up complete on double cast-iron sleeve flanges; to be thoroughly built and turned true, flanges keyed to cam-shaft.
1 Set upper hard wood guides for stems, bored for stems and bolts.
1 Set lower hard wood guides for stems, bored for stems and bolts.
2 Jack-shafts of wrought iron.
4 Side boxes or brackets for jackshafts.
10 Cast-iron sockets for levers.
10 Hard wood levers for same, for stamp holders.

All bolts, rods, nuts and washers for the entire framework of battery according to plans furnished. When the mill is wet crushing one set of water pipes, valves, and fittings for stamps proper. Complete working drawings for setting all the machinery, and plans for the frames also must always be furnished, so that any competent millwright can build the frames and set the machinery. In some cases a knee frame for the battery is used, and the tightener is not then required, a friction clutch pulley often being used to facilitate stopping the battery independently of the power.

LIST OF SIZES—COMPLETE IN DETAIL AS PER SPECIFICATIONS ABOVE.

Weight of Ironwork, including Wood Pulley and Guides.

5	stamps,	450	lb. each	...	...	...	...	9,500	lb. each
10	,,	450	,,	...	...	...	...	18,000	,,
5	,,	550	,,	...	...	...	...	10,000	,,
10	,,	550	,,	...	...	...	...	19,500	,,
5	,,	650	,,	...	...	...	...	11,000	,,
10	,,	650	,,	...	...	...	...	21,500	,,
5	,,	750	,,	...	...	...	...	12,500	,,

10	stamps,	750	lb.	each	...	...	...	...	24,500	lb. each
5	,,	850	,,		...	...	...	...	15,000	,,
10	,,	850	,,		...	...	...	...	29,500	,,
5	,,	950	,,		...	...	...	...	17,000	,,
10	,,	950	,,		...	...	...	...	33,000	,,

Battery Frames.

1	complete	frame	for	5	stamps	for	550	and 650	stamps, weight	9,600 lb
1	,,	,,		5	,,		750	,, 850	,, ,,	12,000 ,,
1	,,	,,		10	,,		550	,, 650	,, ,,	17,000 ,,
1	,,	,,		10	,,		750	,, 850	,, ,,	20,000 ,,

The following Tables give the power required for the mills of different capacity :*

Ten-Stamp Wet-Crushing Gold Mill.

	Horse-power.
One Blake rock-breaker, No. 2...	6
Two ore-feeders	0
Ten stamps, 750 lb., ninety drops	12
Four Frue vanner concentrators	2
One grinding pan, 3 ft. diameter	3
,, settler	3
Friction	4
Total	30

Such a mill is capable of working 15 to 18 tons per day of twenty-four hours.

20-Stamp Wet-Crushing Gold Mill.

	Horse-power.
One Blake rock-breaker, No. 2	6
Four ore-feeders	0
Twenty stamps, 750 lb., ninety drops	23
Eight Frue vanner concentrators	4
One grinding pan, 3 ft. in diameter	3
One settler...	3
Friction	7
Total	46

Such a mill is capable of working 35 to 40 tons of ore per day of twenty-four hours.

Power Required for a 10-Stamp Wet-Crushing Gold Mill.†

	Horse-power.
Two Challenge ore-feeders	0
Ten stamps, 750 lb., ninety drops, 7 in.	12
Five Hendy concentrators, or three Triumph concentrators	3
Friction (20 per cent.)	3
Total	18

* Fraser and Chalmers, Chicago.

† Joshua Hendy's works, San Francisco.

The crushing capacity of the above mill should be from 20 to 25 tons of ordinary gold-bearing quartz per day of twenty-four hours.

Power Required for a 20-Stamp Wet-Crushing Gold Mill.

	Horse-power.
Four Challenge ore-feeders	0
Twenty stamps, 750 lb., ninety drops, 7 in.	24
Ten Hendy concentrators, or six Triumph concentrators ...	6
Friction (20 per cent.)	6
Total	36

The crushing capacity of the above mill should be from 40 to 50 tons of ordinary gold-bearing quartz per day of twenty-four hours.

Power Required for other Appliances of Stamp Mills.

	Horse-power.
Challenge rock-breaker, No. 2	5
,, ,, No. 3	6
,, ,, No. 4	9
Stanford ore-feeder	0
Tullock ore-feeder	0
Howell White furnace, 40 in.	2
Amalgamating pan, 5 ft. in diameter	3
Clean-up pan, small	2
Revolving barrels...	3

With a view to showing the advantages of labour-saving machinery in the treatment of gold-bearing ores, the following is a statement made July, 1876, of the operations of the Rawhide Mill, of twenty stamps, steam-power without them. The mill and mine have not been worked since then.*

Milling Costs—Rawhide Mills, July, 1867.

	$	
Engine :		
One engineer	6.00	
,, ,, ...	4.00	
4½ cords of woods, at $4	18.00	
One fireman and helper	2.50	
Oil, &c....	50	
		$31.00
Cost and Power.		
Stamps :		
One rock-breaker	2.75	
Two battery feeders, at $3	6.00	
Hauling ore	3.50	
Shoes and dies	3.00	
		$15.25
Carried forward		

* Prod. Precious Metals in the U.S., 1883, p. 726.

Brought forward		$46.25
Pans and amalgamators :		
One amalgamator	5.00	
Two pan-men, at $3	6.00	
,, tank-men, at $2.50	5.00	
One assistant	3.50	
Quicksilver	4.00	
Lights, oils, chemicals, &c.	1.50	
Wear and tear of shoes, dies, and pans	7.00	
		$32.00
Add sundry contingencies		1.75
Total		$80.00

Or at the rate of $2 per ton on a working capacity of 2 tons to stamp.

With the addition of superintendence and extraordinary repairs, the cost would be at least $2.50 per ton.

Water Required for Quartz Mills.

The quantity of water required for reducing and amalgamating gold and silver-bearing ores by the wet battery process is generally estimated as follows, viz. :

For boiler, 7½ gallons per horse-power per hour.
,, each stamp, 72 gallons per hour.
,, ,, pan, 120 gallons per hour.
,, ,, settler, 60 gallons per hour.
,, ,, concentrator, 60 gallons per hour.

If the water used in the battery, pans, and settlers be run into settling tanks, it can be re-used with a loss of about 25 per cent.

In any and all of the methods in use for the treatment of gold, there are a certain number of losses of gold which will occur, and which arise from several causes. These are improper treatment, when the tailings will be rich, and the metal lost will be found in them, though it may not be possible to extract it; careless handling, when the lost gold may or may not be found; improper sampling of the ore, when the assays will give too high yield; and the dishonesty of the men. Careless handling can always be guarded against; but it has happened many times that the most profitable work done has been in clean-ups around the foundations, or underneath the platforms of the machinery, especially where mercury is used. The amount of continual loss which arises from dishonesty is very small, and is hardly worth discussing; that such losses do occur occasionally is beyond

doubt; they can only be prevented by watchfulness, as an habitual thief always ends by being careless from impunity, or in some other way attracting attention. The loss from improper sampling will not be a real one. The loss in the tailings is therefore the only one which merits attention. This may arise from several causes; first, improper treatment of the ore; second, from having too much or too little mercury or water, or both, in the mill; or third, by improper arrangement of the mill itself, which is a capital defect.

In determining the loss of gold by the mill process a series of assays were made of the tailings of one of the best mills of California*. The following Table gives the results obtained. Each assay reported in this Table was made in triplicate, and the result given is the mean from the three operations. The samples were procured at the end of the tail sluice, and were taken every half-hour during a twenty-four hours' run. The buckets were allowed to run nearly full, and the sands were settled exactly three minutes. The water holding the slime and float gold in suspension, was filtered off on a large paper filter capable of holding the whole contents of the bucket. The coarser sands, comprising Nos. 2, 3, and 4, were filtered on a separate filter.

ASSAYS OF CLASSIFIED TAILINGS.

No.		Sample 1.	Sample 2.	Average.
		$	$	$
1	Slime, including float gold, representing 19 per cent. of the pulp	12.40	11.66	12.03
2	Slime representing 51 per cent. of the pulp	15.70	14.90	15.30
3	Sand, representing 23 per cent. of the pulp	34.10	22.40	28.25
4	Coarse sand, representing 7 per cent. of the pulp	22.52	40.18	31.35

The rock crushed paid at the time of sampling $47.40 per ton, exclusive of the value of the concentrates.

Estimating the yield of the rock in the mill at $47.40, the sulphurets, or concentrates, at $2.50, the value of tailings as the proportional average of Nos. 2, 3, and 4 per assay at $16.49,

* Report of Mr. G. F. Deetken, Grass Valley, Nevada County.

the value of float gold at $2.28, the total contents of gold in the quartz crushed, was $68.67, of which $18.07, or 27 per cent., was lost in the tailings, 3½ per cent. being float gold. No gold was visible in the above samples.

The losses may arise from the presence of a kind of gold which will not amalgamate, and which the miners call "rusty gold," to distinguish it from that which is caught by the mercury. To clean this coating from the gold has given rise to a great many inventions of machines and patents for chemicals, which are intended to break up the combination between sulphur and gold, on the supposition that the gold is in a state of chemical union with it; or to loosen the cover from the surface, on the supposition that it is only a coating. A number of types of furnaces have been declared to be the only means of getting rid of the rust, and notwithstanding their use, the gold still obstinately remains rusty, or at least is lost. The last cause of loss is the presence of particles of gold so thin and so light that they float on the water, and are carried off on its surface, never having the opportunity of coming in contact with the mercury. The gold may exist in this condition in the ore, for which there is no remedy; but long sluices, which do not generally much more than pay for themselves, catch some of it. Or the gold may be put into this condition by the stamp, in which case the remedy is to be applied in the mortar, either by forcing the discharge, amalgamating in the battery, or both; but notwithstanding all the precautions, if the ore tends to make float, it will make more or less of it, and the effort must be directed to making the least possible amount. What the best method of working the ore should be, ought to be determined before the mill is erected, as, after it is once built, changes can be made, or mistakes remedied, only at very great expense.

There has always been a theory among those working placer mines that gold is both found "rusty," and becomes so under treatment, by which they mean, not that gold becomes coated with oxide of gold, but that it is either coated superficially or alloyed with some substance which prevents the contact with mercury, and thus precludes the possibility of amalgamation. To such conditions of the gold are ascribed the losses in

working most placer deposits and some of the losses in stamp mills. Rusty gold is often talked about, but is not often shown, and when produced oftener amalgamates than not. It is frequently covered with a brownish coating, and has a much redder colour than ordinary gold. The coating is very irregularly distributed over the surface, every spot where the least abrasion has occurred showing the true gold-yellow. All such spots form contact surfaces, and cause the whole piece to amalgamate readily. When the gold is wholly coated it resists the action of the mercury for a considerable time and perhaps altogether. This coating on gold is due to the superficial action of some substance which is soluble and is often precipitated at the same time and by the same causes as the gold, for fine particles of gold are sometimes visible with the microscope in the detached coating. It has never to my knowledge been carefully analysed. It is supposed to be produced by a coating of silica cemented by sesquioxide of iron, produced by the decomposition of the pyrites associated with the ore, or to a partial oxidation of the base metals associated with the gold, and is said to occur both in placer and vein gold. It is easily removed either by rubbing or solution in acids. Those who describe it say that it often cracks off from pieces of gold, leaving them bright. Nuggets of some size are said to have been found in South America, in alluvial soil, coated with a silicate of iron containing considerable gold. Very often the film is entirely composed of silica, which is deposited on and beside the gold. This silica is sometimes opaque, and again quite transparent, so that the gold can be seen, with the microscope, disseminated through it, just as cinnabar crystals are seen in the red chalcedony of the district around Knoxville, California. When the silica surrounds the gold entirely it prevents it from being attacked by the mercury, just as the fine particles of gold from the veins are carried off in the rock when it has not been rendered sufficiently fine in the crushing. While there are many artificial causes which produce the rustiness of gold, this covering of the surface with particles of some foreign substance is probably the only one which occurs in nature. In the separation of gold from its ore there are a number of causes which render it rusty or prevent its amalga-

mation in the mill. Some of these causes I have had occasion to investigate, and some of them are, I believe, new; others have been more or less well known for a number of years.

It has been asserted that the presence of certain substances alloyed with the gold would prevent its amalgamation. This I have generally found not to be the case. I have made a number of these alloys and have found them to amalgamate without difficulty, but chemical combinations frequently do resist the action of the mercury altogether, or when they yield it is attended with the formation of a chemical compound with the mercury, and the consequent loss of the latter.

In order to ascertain the causes which prevent the amalgamation of gold, I undertook to reproduce artificially the conditions which were supposed to prevent it. These I found to be mechanical and chemical. I found that if a piece of soft gold which could be easily bent, and amalgamated readily, was hammered on a perfectly bright anvil, with a bright hammer, striking heavy blows in quick succession, until the gold had become hard, and had acquired a certain amount of elasticity, it would remain in juxtaposition with mercury for a very long time without being affected by it. This hammering increases the density of the metal and closes the pores, so that I have recently had a piece of gold which was put into this condition by hammering, remain nearly two weeks floating on mercury without being attacked. If the gold in this condition is heated and cooled slowly, it again amalgamates rapidly. If, however, it is cooled rapidly by plunging it suddenly into very cold water, the amalgamation takes place very slowly. The same metal, after being repeatedly rapidly cooled and heated, amalgamates more readily than if it has only been treated once.

I have several times renewed the experiments on the non-amalgamation of hardened gold with great care with similar results. When chemicals were used the results were peculiar. A neutral solution of corrosive sublimate* will scarcely attack gold at all under any conditions whether hard or soft. I have repeatedly tried the experiment of placing the pieces of gold in a neutral solution and touching them with a point of iron, such

* Designole process.

as a knife-blade. The blade of the knife became invariably corroded, and amalgamation sometimes took place at the point where the iron touched the gold, but spread no further even after prolonged contact with the iron. When this same soft gold was placed in an acid solution and touched with an iron point, the amalgamation took place instantaneously. When, however, the gold which has been hardened by beating on an anvil is subjected to even the acid solution, it amalgamates very slowly, even with prolonged contact with the salt of mercury.

Samples of pure gold and of gold alloyed with copper were first rolled until they were extremely hard, and then were hammered on an anvil and were placed in contact with mercury. After a considerable time they began to amalgamate very slowly. Samples of the same gold were then annealed in an ordinary Bunsen flame, after which annealing they became perfectly black, and would not amalgamate at all under any conditions, owing to the formation of a copper oxide upon the surface of the metal which prevented contact. When, however; the original alloy, and also the alloy annealed, was heated for a short time in nitric acid, they both amalgamated with equal readiness. It appears therefore that not only will fine gold which is hammered until it is very hard be prevented from amalgamation, but that the ordinary alloys of gold will also be prevented from amalgamation. When fine gold, as previously stated, is hammered, it does not amalgamate; but when it is annealed, it will immediately amalgamate. This appears not to be the case with all alloys, where there is an opportunity in the course of annealing for an oxide to be formed.

In order to reproduce the conditions in which metallic gold is found in the stamp mills, strips and pieces of gold were pounded in a perfectly clean iron mortar with a clean pestle in as nearly as possible the same conditions as they would be in an ordinary stamp mill. It was found that when the stamping was done under water and with light blows, and for only a short time, while there was an evident increase in the difficulty of amalgamating, the gold did amalgamate with sufficient readiness to insure that the most of it would be caught by the mercury after prolonged contact. When, however the blows were very heavy and

the pounding continued for some time so as to flatten out the piece, prolonged contact with the mercury produced only a very slight amalgamation, which did not even show the smallest trace under five minutes, and was only partial even after half an hour. Every attempt was made to make the mercury attack this gold. It was placed directly in and on it, but the quicksilver rolled over and over as if the gold had been so much sheet iron. Even when the mercury commenced to adhere to the gold it was in very minute spots, which did not spread for a long time. When a piece of the same metal was treated with acid, there was a perceptible difference in the rate of amalgamation; but it was still very slow, too slow for the amalgamation of an ordinary stamp mill. When these same pieces were annealed they amalgamated immediately. The same operation was repeated on pieces pounded in a mortar without water. The amalgamation took place even more slowly than before. When the pieces were cleaned with acid they amalgamated about as slowly as the pieces treated under water which had not been cleaned. It took a much longer time to produce the adherence of the fine globules of mercury, and they propagated themselves much more slowly. In every case where a freshly broken edge came in contact with the mercury it was attacked at once, showing that a film had been formed which prevented contact. This film is slowly dissolved by acids and quickly dissipated by heat, for when they were annealed the pieces amalgamated at once. The same results were obtained when the gold was beaten on a dusty anvil.

The same experiments were tried with silver. Pieces obtained from an assay, which were consequently nearly pure, were pounded in a mortar, some with, others without, water. Those in which the water was used presented a perfectly clean appearance somewhat duller than the ordinary colour of silver, and when made into a small dish and a globule of mercury placed upon it, held the mercury for a considerable length of time without being affected by it at all. When the mercury was placed in a vessel, and the silver dipped in it, it did not attack it all at first, but after considerable agitation and rubbing against the sides of the vessel an attachment was made on the edge. When this edge was broken the mercury attacked the fracture at once, very fine

globules being disseminated all over the edge. Notwithstanding this the surface resisted the contact of the mercury for a considerable length of time, and then was only irregularly attacked. When the piece was treated in diluted sulphuric acid for some time and then cleaned and treated with mercury, it resisted in the same way, but for a shorter time. When this piece so cleaned was annealed in a Bunsen burner, the mercury attacked it instantly.

The pieces which had been pounded in the mortar without water, had a much darker appearance in patches unevenly distributed over the surface of the piece. It resisted the action of the mercury for a great length of time, but when placed in a vessel with the mercury and treated with sulphuric acid, it acted in about the same way as the piece pounded under water did before being cleaned. It would have taken a very long time to have amalgamated the whole of the piece. When the cleaned piece was, however, annealed and placed in the mercury, it amalgamated like all the others—instantly.

It thus appears that silver as well as gold is affected by pounding, and that under the most favourable conditions, where no foreign material is present, it will amalgamate with very great difficulty. When, however, any foreign material, as dust, or small particles of ore has been allowed to remain in the mortar or on the pestle, the amalgamation took place even more slowly than before, and in some cases after a number of hours no perceptible effect of the mercury could be observed. That this coating is a superficial one, that amalgamation is possible where there is a freshly broken edge, and that this amalgamation on the edge will eventually extend through the whole piece, and that the pounding of the pieces is undoubtedly a source of loss in the mill, these experiments seem to settle decisively.

It is well known to silversmiths that when silver is annealed after being hardened it turns black, and that when this is placed in the pickling acid the black substance is dissolved upon the surface and it becomes perfectly white. When, however, this white coating is brushed into, there appears below it a purple coating which is composed of the oxides of copper and of silver, and this purple coating cannot be removed except at considerable

expense. A microscopic examination of the surface of the piece showed that when the pickling acid attacks the coating of oxides, it does so only superficially, leaving a white porous surface which is removed in the process of polishing, and leaves a purplish surface, which, when highly magnified, shows the oxide below. As the oxides are lodged in the bottom of the lines of the scratches which are produced by the polishing, the decomposition of the light gives the appearance of a purple colour. When, however, the objects are highly magnified, only the white colour of the pure silver and the brown in the bottom of the depressions of the scratches are seen. It seems probable that similar oxides may form upon gold-alloys and prevent the contact of the mercury.

To ascertain the effect of different substances on the gold, a strip of clean gold which readily amalgamated was dipped into a solution of sulph-hydrate of ammonia, and also into one of sulphuretted hydrogen. After being withdrawn from these liquors and allowed to dry, mercury would not touch either of the pieces. The effects of grease are well known, and the greatest care is taken in most mills to keep the "quick" bright, either by rubbing or by the addition of chemicals, to such an extent in some cases that the chemicals themselves used in excess are often a cause of the very thing they are intended to prevent. All these causes, the hammering, the effect of sulphuretted waters, and also the effect of grease, occur in every mill, and probably have something to do with the losses in gold which take place there.

The effect of the different elements likely to be found with gold was also investigated. A strip of gold which would amalgamate easily was exposed to vapours of sulphur, after which it would not amalgamate. When the piece was heated in the flame of a Bunsen burner, the film of sulphide was decomposed, and the gold amalgamated readily. To test the effects of arsenic and antimony upon gold, these metals, in variable proportions, were melted with it, and it was ascertained that considerable quantities of arsenic and antimony were in the alloys. Both these alloys amalgamate very easily. Gold and arsenic and gold and antimony were then precipitated together, and it was found that the precipitates were compounds of antimony and gold, and of arsenic

and gold; they also amalgamated readily. It has been asserted that .0004 gramme of antimony to the ton of auriferous pyrites of Grass Valley, and .001 gramme in that of Callao, was sufficient to prevent the amalgamation of the gold.* This seems, however, to be not quite exact, if the statement given of the conditions in which the gold is found be correct, for there does not seem to be any reason why an alloy with a much larger amount, as in the artificial compounds, should amalgamate readily, and the natural one, containing so much less, refuse to do so.

The effect of sulphur was further studied by making a sulphide of gold. When this was heated it was found that the sulphur was readily driven off by ignition, and the residue, which was bright yellow gold of exactly the same shape as the sulphide, amalgamated easily. A phosphide of gold was then made by pouring melted phosphorus upon hot sponge-gold, and expelling the excess of phosphorus by heating for some time in a non-oxidising atmosphere. It did not amalgamate. Oxide of gold was made, but this compound is so very unstable and so readily reduced to metallic gold by simple rubbing that it could not possibly have any effect on preventing amalgamation, although it is not attacked by mercury. An amalgam of gold was then placed in acid, which easily dissolved out the mercury, and left behind a brown crystalline residue of gold, which readily took up mercury again.

I also made a series of experiments which consisted in fusing mispickel free from gold with a certain amount of metallic gold, and experimenting with that; also fusing the same mineral with sulphide of antimony, and fusing iron pyrites with gold and compounds of antimony. The mixtures were made with mispickel taken from a vein not far from Stamford, Conn., absolutely free from gold. It was fused with gold in a crucible, reduced to powder, carefully roasted, sampled, and assayed, and found to contain 143 oz. of gold to the ton. This mixture was then amalgamated in an iron vessel without chemicals with clean mercury, and by this method 36 oz., or 25 per cent., were extracted, and 107 oz. were left in the tails. It was then amalgamated with a neutral solution of bichloride of mercury in an

* Comptes Rendus, March 17th, 1879.

iron vessel and 11 oz., or 7.5 per cent., were extracted and 132 retained in the tails, making a total of 32.5 per cent. extracted.

Another sample was treated with mercury and an acid solution of bichloride of mercury in an iron vessel, and 13 oz., or 9 per cent., were extracted and 130 left in the tails, the acidity of the solution having produced no effect. An electric current was then passed through a sulphuric acid solution with mercury; 23 oz., or 16 per cent., were extracted and 120 oz. left in the tails, or 25 per cent. altogether extracted.

A fresh sample was then treated with cyanide of potassium and mercury in an iron vessel, and 62 oz., or 43.3 per cent., were extracted while 81 oz. were left in the tails. These tails were then re-amalgamated with cyanide of potassium; 2 oz. were extracted and 79 oz. left in the tails. These tails were again re-amalgamated with salt, bichloride of mercury, and mercury and 2 oz., or in all 46 per cent. were extracted and 77 oz. were left in the tails.

After repeating these experiments a number of times the conclusion was drawn that on artificially prepared substances such as these, made by fusion, the most careful and prolonged roasting did not sufficiently destroy the combinations of the gold, or more probably the coating of oxide of iron on the gold, so as to enable the mercury to reach it even when bichloride of mercury was added with a neutral solution, and no advantage was gained by treating such an ore with bichloride of mercury, even in an acid solution; and that even with cyanide of potassium, which acted the most powerfully of all the chemicals, the amount of loss would necessarily be very large, and that it is consequently dangerous in roasting ores of gold containing sulphur and arsenic ever to allow a high temperature in the furnace. The amount of sulphur contained in the roasted ore was not determined chemically, but it was quite small, and so was also the amount of arsenic. The roasting was prolonged and very carefully done, every precaution being taken to prevent such an elevation of the temperature as would cause the ore to fritt, and every effort was made to have the processes conducted exactly as they would have been in a mill.

Sulphide of antimony free from gangue was then mixed with

the pulverised arseno-pyrite, and fine sponge gold added to it, and the whole fused in a crucible. It was then pulverised, roasted, and assayed, showing $516.72 to the ton. It was than treated in an iron vessel with cyanide of potassium and mercury. The tails assayed $69.27, showing that 88 per cent. of the gold had been extracted.

Another sample of the same ingredients was melted in a crucible and assayed 249.5 oz. to the ton after amalgamation. The tails contained 57.7 oz., showing that only 76.9 per cent. of the gold had been extracted.

Pulverised iron pyrites, stibnite, and gold were then fused together; carefully roasted it assayed 56 oz. to the ton. The roasted ore was then amalgamated with cyanide of potassium in an iron vessel, and 9.1 oz. were then extracted. The tails were then amalgamated with bichloride of mercury and mercury in an iron vessel, and 4.8 oz. extracted, or 13.9 oz. altogether, showing an extraction of only 24.82 per cent. As a miscroscopic examination showed a very few exceedingly small spots of undecomposed pyrites, it was thought best to repeat the experiment. Another mixture was made which, after roasting, assayed 120 oz. to the ton. The roasting was done with the greatest care at a low temperature. After the fumes ceased to be given off, powdered charcoal was added to reduce any oxidised compounds. The excess of charcoal was burned out at as low a temperature as possible. Only 25.36 oz. were extracted, or 21.14 per cent., thus confirming the previous experiment.

These experiments were repeated on mispickel containing gold. The raw ore contained 14.17 per cent. of sulphur and 20.50 per cent. of arsenic. The roasted ore contained 0.67 per cent. of sulphur and 1.87 per cent. of arsenic. The assay of the roasted ore showed it to contain $67.88 per ton in gold and $5.17 in silver. On treating it with cyanide of potassium and mercury in an iron vessel the tails were found to contain $14.16 of gold and $3.67 silver, so that 75 per cent. of the gold, and only 30 per cent. of the silver, was obtained by this process. This treatment of the roasted ore was much more successful than with the material which had been fused, and shows the necessity of avoiding high temperatures in roasting gold ores.

It is evident that under conditions artificially produced without the use of expensive chemicals, there is something which prevents the amalgamation when arsenic, antimony, and sulphur, are together in the ore. Exactly what this is I have not yet been able to ascertain, but hope at some future time to be able to discover exactly what the conditions are, which when sulphur antimony, and arsenic are present, prevent the gold from being attacked by the mercury; but I have little doubt that it is for the most part a coating formed during the roasting, and that the imperfection of the trials is owing to the same cause, and to the very small quantities necessarily used in the laboratory tests.

It will thus be seen that the stamp-mill is not a rationally designed machine. The action of pounding is likely to put some of the gold into such a condition that the mercury will not touch it, and to flour the gold, as well as the quicksilver. There is, besides, in the mill every probability of the introduction of grease or greasy substances, like the powdered hydrated silicates of magnesia and of alumina, which not only froth but coat the gold with a slime which prevents the action of the mercury. If the water used in the mill is not pure, there is a further likelihood of the introduction of sulphuretted hydrogen, and of other soluble sulphides, which act superficially on the small particles of gold, and prevent the action of the mercury. I have elsewhere pointed out some of the other causes of losses in the treatment of gold. The pounding action of the stamp has doubtless the effect to make some of the particles of gold wholly unattackable by mercury, while the action of rubbing that occurs in the arrastra is much more likely to pulverise the fine particles of pyrites, to break up any coating that may be around the particles of gold, and to rub off any superficial deposit, and thus bring the gold into contact with the mercury and make it amalgamate. It would be folly to advocate the reintroduction of the arrastra, which is so limited in its output, but I am satisfied that we shall have to come back to its principle. It is a very remarkable fact that when, "in the early days," Mexicans with the arrastra got $50 to $60 a day, the stamp-mill working on the same rock obtained only $15 to $20, and instances are cited where with the best modern machinery only $20 to $30 can be

got out of rock which yields $700 to $800 by the fire assay. One or two machines have recently been constructed on the arrastra principle, but no systematic tests have been made of them, and little done beyond the singing of their praises by enthusiastic inventors.

CHAPTER VI.

PLATTNER'S PROCESS AND LEACHING GOLD ORES CONTAINING SILVER.

PLATTNER'S process for the extraction of gold by chlorine gas was introduced into the United States in the year 1858 by Mr. G. F. Deetken, in the Eureka and Idaho Mills, of Grass Valley, California, for the treatment of the concentrates from the various gold mines of that region. From its successful use in that place the process was generally adopted in the other gold mining regions of California for the treatment of concentrates, but it was not until the year 1863 that it became an important factor in the production of gold. Grass Valley has been for many years famous on account of the Idaho gold mine, which, with the other mines in its vicinity, have made it well known as a gold mining region. In this district there are many veins of quartz which are intersected by beds of both slate and granite. A part of the gold found in these veins is free milling, and a part of it is contained in the pyrites which is found in and concentrated from the tails, after as much of the gold as possible has been separated by battery, blanket, plate, and sluice amalgamation.

The percentage of free gold in these various mines is extremely variable. In some of them, as the Idaho Mine, the largest proportion of the gold is free milling, while at many of the other mines in that vicinity and elsewhere the ores contain large quantities of sulphurets. In a few instances they will amount to as much as from 7 to 10 per cent. of the total volume of the ore, while in most others they will not be more than from 1½ to 2 per cent. Their value is generally from $40 to $100 to the ton. The Table on the next page gives the values at some of the principal mines.

PERCENTAGE AND VALUE OF SULPHURETS.

Name of Mine.	Per Cent.	Value in Dollars to the Ton.	Name of Mine.	Per Cent.	Value in Dollars to the Ton.
Stickels . .	2	120	Keltz . .	2	200
Utica . . .	2	60	Buchannan .	1	120 to 260
Plymouth . .	1½	135	Idaho . .	½₀ to ½	80
Bunker Hill .	2	57	North Star .	2 to 3	100
Keystone . .	1½ to 1¾	110	Empire . .	2	150 to 300
Zeile . . .	2¼	100	Providence .	5 to 6	130
Stewart . .	¼ to 1	60	Nevada . .	2 to 3	100
Moore . .	2	130	Delhi . .	½ to 3	80 to 100
Mamouth .	4	150	Eagle Bird .	¾ to 1	100
South Springhill	1	125	Sierra Buttes .	½	90 to 550
Abbey . .	2½	225			

They consist mostly of iron pyrites, but as all that is heavy in the tails is caught, the concentrates may contain the sulphides of other metals if they were present in the ore, so that not infrequently besides the pyrites, copper pyrites, galena, and arseno-pyrite will be found in them. When the ores of certain districts of both Colorado and California contain tellurium, this will also be present and make them difficult to treat. These substances do not, however, prevent the application of the process unless the base metals other than iron are present in very large proportion. Concentrates are occasionally allowed to weather from three months to a year, salt sometimes being added to assist the decomposition, when it is the custom to work them in pans.

The ores are generally crushed fine enough to pass a 40-mesh screen. The method of crushing the rock and of amalgamating the free gold and some of the methods of concentrating the tailings have been described.* Of all these methods the use of the Frue vanner has of late years been one of the most successful. It has been very largely introduced in all the gold mines of the West. Every battery of five stamps is generally provided with from one to two of these machines. When they were first adopted it was supposed that one machine would be sufficient. It has been found, however, that there is a great saving in using two. In this case the ore is amalgamated in the battery and on

* Vol. i., chap. iv.; vol. ii., chap. v.

copper plates before passing over the machines. There are very few ores upon which Plattner's process can be applied directly. It is almost exclusively applicable in this country to the gold, which is not free, and consequently is not caught by the mercury in the various systems of amalgamation, and has become concentrated in the tails. There are, however, some ores and many concentrates to which the process cannot be applied.

The principle of Plattner's process consists in attacking the gold in the ore by chlorine in order to form a soluble gold chloride, which is afterwards dissolved out with water. Every possible variation has been given to this simple statement of the process. It has been patented under many forms, using nascent chlorine in contact with the ore, using it under pressure, using it in the condition of bi-chloride of mercury; but no process has been so extensively used and, in the long run, so successful as the simple one invented by Plattner, and applied by him, of bringing chlorine in the condition of gas in contact with the gold contained in the ore. In order to apply it most effectually there must be no material present which will be attacked by the chlorine except the gold, which must be either in a metallic state as it comes from the mine or its combinations or coatings must be destroyed by roasting.

The chlorine itself must be purified from everything that would be likely to cause a reaction between it and the other constituents of the ore, or between any of the constituents of the ore themselves. In the various chlorination works the pulp which is to be subjected to treatment comes from the mills in a very fine state of division, and in custom works or in those which purchase concentrates no machinery for crushing ore is usually associated with the works. The only precaution necessary to take with the concentrates, as they are called, after they are delivered to the mill, is to see that they are kept wet until they are ready to go to the drying floors just previous to their introduction into the roasting furnace. If they are allowed to dry the decomposition of the pyrites commences, which forms lumps which must be crushed before roasting, as they are not easily broken up in the furnaces. The oxidation of the iron may cover the gold with a coating, which is not readily removed, and from such material

the sulphur is not all driven off. Some of the precious metal in such cases will not be attacked, and will consequently be lost in the tails.

In giving the description of this process it is not the intention to describe any particular establishment. The details for the whole process will be given, as they vary in different works. They have been taken from the chlorination works in Alaska, those now and formerly existing in Calaveras, and Amador, Nevada Counties, California. There were formerly a large number of works in and about Grass Valley which have been abandoned. Many of them were too small to be of any importance. There are three classes of these works—those which purchase ores for treatment on their own account, mills which treat their own ores, and custom mills which treat them for the owners of the ores. In the latter case each customer's ore is put by itself in bins, and is kept carefully separate from the other ores. In the first case, no special care is taken with the ore except to have it previously assayed. It is not generally considered advantageous to treat ores poorer than $25 per ton, but ores as low as $15 and sometimes lower are occasionally purchased at a very low price and then mixed with richer ores. The sulphurets treated, as shown by the Table, page 597, contain extremely variable amounts of gold. They are all in a very fine condition, having come from the preliminary treatment in the mills. Only ores with a completely insoluble gangue can be advantageously treated. Of these, all are not equally well suited to this process, for if the gold is in coarse particles the action of the chlorine would be so slow, acting only from the outside, that it would take so long a time to convert it all into chloride, that the process would become impracticable. For this reason ores containing coarse gold cannot be well treated by the chlorination process. There are two distinct types of the process, the vat and the barrel process. The vat process was the one invented by Plattner. The barrel process has grown up within a few years.

The Vat Process.

As all the ores are sulphurets this process consists of five different phases:

1. Roasting the ore.
2. Impregnating the roasted ore with chlorine.
3. Leaching the impregnated ore.
4. Precipitating and fusing the gold.
5. Treatment of the leached tails for silver.

1. *Roasting the Ore.*—The tails from the amalgamation mills are composed for the most part of the combinations of gold and silver with sulphur, arsenic, iron, or copper. The average composition of the tails is given below.*

ANALYSES OF TAILINGS.

	Eureka and Idaho Mines, Grass Valley.	Washington Mine, Mariposa County.	Black Bear Mine, Klamath County.
Copper	0.85	0.00	0.00
Lead	0.78	1.50	0.00
Gold	0.02743	0.00914	0.0137
Silver	0.0068	0.0035	0.003
Zinc	0.00	1.34	0.00
Iron	40.65	30.85	42.05
Arsenic	trace	0.00	21.25
Sulphur	32.80	31.33	25.10
Silica	12.64	33.30	10.35
Alumina	0.10	0.00	0.85
Magnesia	3.50	0.00	0.00
Oxygen and loss by differ.	8.65	1.67	0.38
	100.00	100.00	100.00

The object of the roasting is to convert all the metals, except the gold, into oxides, so that they will not be attacked by the chlorine, leaving only the gold to be acted on. This transformation into oxide is done to prevent the formation of soluble salts of the base metals which would precipitate the gold in the leaching vat. The impurities most to be avoided are, sulphur, antimony, and arsenic, since their presence would cause the precipitation of the gold. Lead, lime, and magnesia are very deleterious, as they are likely to be attacked by the chlorine. It is also necessary during the roasting to avoid the production of any soluble iron chloride or any other soluble metallic salts, as they will act on the iron oxide, and also precipitate the gold once dis-

* Mineral Resources of United States West of Rocky Mountains, 1872, page 337.

solved, and leave it in the tails, in the ore tanks, after lixiviation. In many districts the gangue of the ore is quartz and contains nothing which would be acted on by the chlorine, but all the rocks in Grass Valley contain more or less lime and magnesia, and therefore absorb more or less chlorine. In order to counteract this as far as possible a small amount of salt is added towards the close of the roasting after the ore has been completely oxidised. This must be carefully done. When the ore is free from impurities in contact with the air and salt, sulphurous and then sulphuric acid are formed in the furnace, which act on the soda and set the chloride free, and this acts upon the gold. When lead is present the salt acts on the lead sulphate, producing sodium sulphate, and a more or less soluble lead chlorides, and oxy-chlorides. If the salt were not added these chlorides and the oxy-chlorides would be formed at the expense of the chlorine.

The extraction of gold by amalgamation from gold-bearing silver ores has always been very incomplete, and is more unsatisfactory according to the length of time during which the ore has been roasted, and is worse the higher the heat. In the Murphy Mill, Ophir Cañon, Nevada, where the ore was roasted in a reveberatory furnace, it was only 25 per cent. When a Stetefeldt furnace was used about 80 per cent. was extracted. At the Alice Mine, Butte, Montana, where the Howell furnace was used, the percentage obtained from January 1 to August 6, 1879, was only 55.6 per cent. At the Lexington Mill, in 1882, only 58.8 and 60.8 per cent. was extracted. As all of these ores contained some silver, there is no doubt that the loss in gold was occasioned by the elevated temperature, necessary to get a high chloruration for the silver. A careful examination made of this subject showed that the gold chloride is very volatile, and that the loss may be from 40 to 90 per cent. of the gold contained in the ore, depending upon the temperature and the length of the operation. The presence of copper chlorides brings it up to a maximum.* When the copper is thoroughly oxidised before adding the salt, so that the amount of copper chlorides is reduced to a minimum, the loss is very much less. There is every reason, however, why the salt should not be added until after a thorough oxidation of the ore

* Trans. Inst. Min. Eng., vol. xiv., p. 339.

so as to prevent the formation of copper chlorides. The losses are also incurred when the ore contains tellurium and selenium.

In making some experiments on gold ores containing copper from Mexico,* these losses were always found to take place when the ores were roasted with salt either in a muffle or a reverberatory furnace. These results were so entirely unexpected that it was at first concluded that an error had been made in the assays. Such was not the case, and in repeating the experiments, by roasting in a muffle, the facts were established beyond any doubt. In the latter case, well-mixed samples were carefully assayed, and then one assay-ton mixed with 3 per cent. salt, and roasted for one hour.

CHARGES ROASTED IN THE REVERBERATORY FURNACE.

Number of Charge.	Gold Value before Chlorurising-Roasting.	Gold Value after Chlorurising-Roasting.	Gold Loss, in per Cent.
2	0.916	0.426	53.5
4	0.262	0.150	42.8
5	0.538	0.075	86.1
6	0.650	0.075	88.5
	CHARGES ROASTED IN THE MUFFLE.		
1	0.700	0.050	93.0
2	0.525	0.075	85.7
3	0.650	0.065	90.0

There is no doubt that the volatilisation of the gold takes place with that of the cuprous chloride. It is increased with the temperature and the quantity of cuprous chloride formed. The loss of the gold is less when the salt is added with the ore, because part of the chlorine acts on the sulphur, which is really only equivalent to a waste of salt, so that there is less chlorine to act upon the gold. The loss is also less when the salt is added on the last hearth, because when the gold is volatilised as chloride there, it is in part condensed by coming in contact with the colder ore over which it must pass to reach the flue. At 100 deg. Cent.† gold is only very slightly volatile in an atmosphere of chlorine. Above this temperature it slightly increases to its maximum,

* Trans. Am. Inst. Min. Eng., vol. xiv., p. 336.

† Trans. Inst. Min. Eng., vol. xvii., p. 26.

which is at about 250 deg. Below a red heat it diminishes to the minimum. It then increases slowly to another maximum at a temperature above melting, and increases continuously between the red and the white heat. The ratio of loss is different at different temperatures. At incipient redness it is .05 per cent. in half an hour. At a low red heat it is .1 per cent., at incipient yellow it is .8 per cent., at melting it is 1.5 per cent., which shows that so far as the gold loss is concerned, low temperature is more important than time in reducing the loss. The loss is also greater with the increase in the quantity of chlorine. It is also greater with the increase in the surface of the gold exposed, and hence will vary with the condition of the gold in the different sulphurets, according as the gold in the pyrites is in more or less minute particles, which accounts for many of the losses in different ores under apparently exactly the same conditions. Of the volatilised material, that found in the condensation chambers is not richer than the ore, which shows that the gold volatilised was in a much more volatile form than ordinary gold chloride. To condense this it is only necessary to destroy the combination by furnishing something to take up the chlorine, which can easily be done by using the sulphurous acid given off from the roasting furnace. A tower filled with coke, through which a small stream of water flows, is well adapted for this purpose.* The gold is then recovered from the coke. Old iron answers equally well, but is more expensive, but there is always plenty of old iron around a mill, and it may, under some conditions, be cheaper than coke. The water coming through these towers is circulated until it becomes too acid. Usually coke is the cheapest material to use. A tower could be built of red brick, laid in a mixture of sand and tar, and lined with firebricks which have been boiled in a mixture of asphaltum and coal tar, so that hot water will not cause it to run. The bricks may be laid up with hot asphalte, and thus be made both gas and acid tight, or the tower may be built of wood painted as the leaching tubs are.

The tower should be so constructed that the coke or whatever material is used for condensation may be charged at the

* *Ibid.*, page 37.

top and withdrawn from the bottom. To work successfully, an exhaust like those used for the fumes of the mercury furnaces in New Almaden, California, should be used.

If the gold contained in the ore is of very low fineness or combined with silver, a large part of it may be lost from a different cause, as the silver chloride will surround the particles of gold and prevent any further action of the chlorine on it. For such cases as these many plans have been proposed in Europe, none of which have to my knowledge been successfully carried out in this country. They generally consist in the use of a strong solution of salt saturated with chlorine. The chlorine attacks the metals, and the salt dissolves them so that they can be precipitated from the solutions. Since the introduction of the lixiviation process in the West, however, a double process is used, which consists first in treating the ore by Plattner's process to get out as much gold as possible, and then treating the tails with a hyposulphite solution, which extracts the remaining gold and all of the silver.

In such cases it will be necessary to chlorurise the silver in the furnace, by adding a few pounds of finely ground salt to the ore, taking precautions not to raise the heat so high as to volatilise any gold chloride formed. This requires very careful management when copper is present. The silver is then leached, the leached ore is carefully washed, and the tails chlorinated and leached for gold, or, better the gold leaching is done first and the silver leaching afterwards. To effect the chloruration, it is generally best to completely oxidise the ore first by a prolonged roasting at a dull red ending with a cherry red, then to add from 40 lb. to 50 lb. of salt for every 100 oz. of silver contained in the ore, lowering the temperature a little for the purpose, and rabble it entirely through the ore. The doors of the furnace are then closed until the salt ceases to crackle. It is then rabbled, and the charge drawn, in from twenty to thirty minutes after the introduction of the salt. It is not safe to add the whole of the salt when the charge of ore is made; 3 lb. or 4 lb. of salt will aid the chloruration of the silver, but more will be apt to cause a loss in gold. It is not safe to increase the temperature above a red heat; even a slight increase in the temperature will cause an important loss in the gold. Exactly

how much it is best to add in each part of the operation can only be told by experience, and careful trials should be made on every ore until the best results are obtained. There is always a decided advantage in leaving the ore in heaps for an hour or more before it is carried to the cooling floor. The chlorine gas is always introduced on the cold roasted ore, and after the gold is washed out the sodium or calcium hyposulphite solutions are used to leach out the silver. It is not necessary with ordinary ores to discharge the tub for this purpose, though it must be done when the ore contains much lead. The launders used for the two solutions should be separate and the reception vats at opposite extremities of the line of tubs. It is not necessary that the extraction of the gold should be complete, as the calcium hyposulphite will dissolve what remains behind with the silver.* When there is a large quantity of lead present the roasted ore is washed clean with water to remove the soluble chlorides and then leached for silver. The leached ore is then removed from the vat, dried until it is as moist as is required for chlorination, chlorinated, and leached, and if it is still rich in silver, treated again with a hyposulphite solution.

The roasting of these ores, as they are all fine, is usually effected in some kind of reverberatory furnace built of common brick. All the different kinds of mechanical furnaces have been proposed for this purpose, and have been used in other countries, but while they are occasionally used they have not, so far as I know, been used to any great extent in the United States, though there does not seem to be any reason why they should not be, except that the limited capacity of the works does not usually justify the erection of furnaces where steam engines have to be used, when the work can be done by men. If the quantity of ore to be treated was very large, there is no doubt but that the use of a mechanical furnace would be much more advantageous than that of a reverberatory furnace. Three different kinds of the latter furnace are used, which vary only in the disposition of the hearths. The hearth may be single and short. A series of hearths may be placed one over the other, the hearths may be single and long, and in the same plane, or they may be

* Trans. Am. Inst. Min. Eng., vol. xii., p. 41.

arranged in steps. The first variety is wasteful of fuel, and there is always danger that the ore will be sintered in it. The second, however arranged, although economical in fuel, is inconvenient. Whenever it is used the doors should be on opposite sides of the furnace and the ground at such a level that they can always be reached. When the doors are on the same side of the furnace, and are reached by platforms, these are always in the way, and prevent the proper treatment on both hearths. The third and fourth varieties are on the same general principle, and are the most economical. The fourth is a little more expensive to build ; but there is a great gain in keeping the ore in the different stages of roasting on different hearths, and then dropping it from one hearth to the other, but when the distance which the ore falls is considerable, the drops or chimneys should be so arranged that the ore in falling does not dust too much, and there be no danger of fresh or unfinished ore dropping on to the hearths below. The last of these furnaces is much in use in California, and the last two are very advantageous both in economy of fuel and labour. It is not generally wise to make very large charges. It has been found best to use for the reverberatory from 10 lb. to 12 lb. for every square foot of hearth as the correct charge, which will usually amount to about a ton on each hearth. The furnace should be fired up several hours before being charged with the dry sulphurets, and the heat should at first be of a degree just sufficient to ignite the sulphur after the material has been evenly spread over the hearth. Too great a heat will matte the easily fusible sulphides, thereby necessitating a withdrawal of the charge from the furnace and a repulverisation. The presence of galena requires a low temperature and continual stirring.

The ore from the Treadwell Mine contains from $5 to $6, partly as fine gold and partly as pyrites. The mill tails are pyrites. They are concentrated on Frues and are chlorinated. The roasting is done in four Spence furnaces with four hearths. They roast 8 tons a day. The ore remains sixteen hours in the furnace. It is brought to the top of the furnace to dry and is charged mechanically from a hopper. On the hearth next the lower one 3 per cent. of salt is added with a special spoon. The

motion to the ore, which was formerly given by fixed rakes, is now done by oscillating ones, which tip every three minutes. These rakes on the upper hearths last a very long time; they, however, last only three months in the lower hearths. There is but little difficulty with them on the upper hearths, but on the lower ones, where the heat must be great to drive off the last traces of sulphur, it is so high that they last but a short time. Formerly the rakes stopped between the forward and the backward motion just over the flue. Now they are made to go beyond it and are no longer exposed there, to the hot sulphurous gases, which is a great improvement. The rails are of steel, but after a time they become oxidised and brittle, and give out, which causes the rakes to get off the track, which breaks them. The rakes work oftener with a jerk than with a steady motion. Formerly Bruckner's cylinders were used, of which there were four; two 9 ft. in diameter and two 8 ft. in diameter and 14 ft. long. They were thrown out on account of the quantity of dust that they made, but otherwise answered better than the Spence.

At Suter Creek, Amador County, California, the furnace is a Fortschaufellungsofen; the hearth is 12 ft. wide and 80 ft. long, inclined from the flue towards the fireplace without any break. It has twelve doors in the side for working the charge, and is capable of working three charges of 2400 lb. to 2600 lb. in twenty-four hours. These charges are worked in shifts of eight hours, one charge is drawn and a new one added every shift. The distribution of the charges on this hearth, however, is as if the furnace contained three charges, which are kept separate with the very greatest care. The one nearest the flue, where the ore is quite cool, is used for drying. The middle hearth is used for burning off the greater part of the sulphur, and here the ore is spread out very thin and is constantly rabbled. The last hearth nearest the fireplace is at as high a temperature as it is possible to raise the ore without sintering it, and here a dead roast is effected. If the temperature should be so high that any of the ore became sintered, it would have to be recrushed and reroasted. It may be said in general that the smaller the charge treated the better the work will be

done, so that it is usual to limit the charge on each hearth to from 1800 lb. to 3000 lb. Before being charged into the furnace the concentrates must be dried, not completely but so that they are no longer moist, and any lumps they may contain broken up by working the ore over by hand. It is also essential that in moving the charges from one hearth to another they do not become mixed, as that lengthens the working considerably. A very low heat is required near the flue, especially if the ore contains lead. The colour of the heat should be a dark red when the ore is stirred, but black on the surface in the daytime; at night it shows a dull red. Constant rabbling on this hearth is not necessary, but the ore should be moved at intervals of ten to fifteen minutes, when the upper surface becomes dull; towards the end only should the temperature be high. Tests can then be made to ascertain whether all the iron and copper sulphate has been decomposed, but generally this is not done, as the men know by experience when the ore is finished. When the flame of the burning sulphur has almost disappeared in the burning-off compartment, ¾ per cent. of salt is added and rabbled into the charge. One or two per cent. of salt added before roasting facilitates the formation of the gold terchloride and reduces the quantity of chlorine gas required. That gold terchloride can be made by roasting sulphurets containing gold with salt is a well-established fact; but although at one stage of the process most of the gold is in the form of terchloride, upon cooling this compound is partially decomposed, a portion only of the gold remaining combined with chlorine. In making the chlorination assay of roasted silver ores containing gold it has been noticed that a portion of the gold is always leached through the filter, showing the presence of terchloride. If any amount of copper is present, a high heat is required to decompose the copper sulphate. If when the furnace is discharged any of the ore should have become lumpy it must be recrushed and reroasted in order to be sure that it is completely oxidised. In some works the finished charge is panned to get the colour of pure gold and to see whether there are any undecomposed pyrites present, but this is not usual. In order to keep a perfect control over the roasting, the ore from each shift is kept by itself and chlorinated alone.

Heywood's Chlorination Works are situated at Angel's Camp, Calaveras County, California. They have a capacity of 3 tons a day. They treat the concentrates from the surrounding mines. Their value is from $40 to $60 to the ton. They are roasted in a reverberatory furnace 80 ft. long and 13 ft. wide. On the inside the hearth is 72 ft. by 11 ft. The roof of the furnace is 13 in. above the bridge, the hearth 12 in., and the grate bars 14 in. below it. The ashpit is 18 in. by 22 in. It has nine working doors on each side. The charge is dried on the top of the furnace, and is from 2100 lb. to 2200 lb. The furnace is worked in three shifts of eight hours each. One charge is drawn per shift. There are always three charges at a time in the furnace. The hearths are called the first, second, and third ovens. No. 1 is the hearth nearest the flue. When the charge is half done, or at the end of four hours, 26 lb. of salt is added on hearth No. 2 and rabbled into the charge. When the charge is on the cooling floor it is dampened down, but not enough water added to make it wet. When pressed in the hand the mass must fall to pieces with the slightest touch. One and a half cords of wood in twenty-four hours are used, and one man per shift works the furnace.

At the Zeile Mill, near Jackson, Amador County, the concentrates from the mine are treated in their own mill. The concentrates are worth $100 to the ton. They are roasted in a reverberatory. At one end of the works there is a store-room for the ore. This is connected with a platform just over the end furnace and a few feet above it, on which the ore is spread out to dry. From a hopper in the centre of the floor it is charged into the furnace below. The concentrates are first weighed in the store-room and the exact charge drawn to the platform. The furnace is divided into three compartments, and has a slight inclination and drop between each. The first division is called the drying compartment. It is 18 ft. long and has two doors. The next is also 18 ft. long and has three doors. The next is 8 ft. to the mill discharging door, and 23 ft. to the end. The ore discharges into a cub under the furnace. The salt is added in the first door from the end of the second compartment, the charge being 56 lb. to 75 lb., according to circumstances. One

ton is roasted in each shift of eight hours. There is but one man on a shift. One and a half cords of wood is used in twenty-four hours, or half a cord to the ton. The wood costs $5 a cord.

At the Amador Reduction Works, near Amador City, they treat concentrates which are purchased in the open market. The roasting is done in a reverberatory furnace 75 ft. long and 12 ft. on the inside and 14 ft. on the outside, with six working doors on each side. There are two drawing doors on the same side as the fireplace where the last roasting is done. The finished charge is discharged into iron cars in the vaults underneath. The end of the furnace next the flue is covered with a staging and the ore is charged from here into the furnace. The doors next the flue are called the drying doors. The ore charged is spread out over a space in the furnace represented by two doors on the outside. After a time it is brought down to No. 3 and spread out and 1 per cent. of salt added. It is then worked forward to No. 2. and worked to the fireplace from there. The ore is dead roasted, The furnaces can treat three batches of 2240 lb. of damp ore, and require but one man per eight-hour shift, and one man in a ten-hour shift to bring wood, fill the chlorination vats, &c. They burn only one cord of wood in twenty-four hours. In front of the furnace there is a platform 15 ft. wide made of wood; 2 ft. below this is a second floor, 12 ft. wide, on which the ore is cooled.

At the Plymouth Chlorination Works, Amador County, they treat the concentrates from their own mills. They are roasted in a reverberatory furnace 80 ft. long and 14 ft. wide on the outside, with a plane hearth without separations. The workmen make three divisions, however, corresponding to the doors on the outside, which they call the "drying," "turning," and "crushing" compartments, and the ore on each of them is kept entirely separate. At the charging end there is the usual platform for charging and drying the concentrates. In the middle or turning compartment the concentrates are spread out very thin, so as to occupy about double the space of either of the other two Here ¾ per cent. of salt is added. There are two drying and six roasting doors, two of them being drawing doors. The usual

work of the furnace is three shifts, one man only on a shift, in which they worked 72 cwt. When they were hurried they worked four shifts of 96 cwt. The furnace occupies the long side of the building; 20 ft. from it and parallel to it is a brick cooling floor 80 ft. wide and 50 ft. long.

At the Lincoln Works, Amador County, California, there formerly was a three-hearth furnace arranged in the shape of steps, each hearth being placed 7 ft. below the one above it. The discharge for the ore is made through a flue the width of the hearth. Each of the hearths is 16 ft. long and 15 ft. wide, and is of the same width throughout. The upper hearth has one door only on the side and one at the end; the middle one has two on the side and one at the end; and the lower one has two on the side only. The drop flue is 12 ft. wide and 12 in. long. In it dampers are placed to regulate the draught, and a door on the upper level for the greater convenience of drawing down the charge. The grate is 8 ft. wide and 16 in. long, and only 4 in. below the hearth line. The height of the middle is 2 ft. in the centre, and 1 ft. at the sides. The fire bridge is only one brick thick. The top is 8 in. above the hearth, and 12 in. to 14 in. above the grate. On one side of this hearth there is a door for discharging the roasted ore into a vault or cub beside the furnace. This cub is 4 ft. by 6 ft., and opens towards the cooling floor. Rails are placed over it for the ore cars to run on, so that the men can conveniently throw the ore into the car. The ore comes from the mine damp, and is placed on the roof of the furnace to dry, and is charged from the roof through cast-iron openings 8 in. square, which are closed with a piece of cast iron and covered with ore. As the ore falls into the furnace dry, the dampers should be closed, and dust chambers should be connected with each furnace. The charge is from 5½ to 6 tons; 2½ to 3 tons of dried ore are charged on the upper hearth, 1½ to 2 tons on the middle one, and 1 ton on the lower one, so that the furnace works from 1 to 1¼ tons per day, varying with the quantity of lead or other fusible substances in the ore. There are generally two men who work ten hours each, and three men working eight hours at each furnace. About two cords of

wood, costing from $5.00 to $5.50 per cord, are used in twenty-four hours.

At the Eureka Works,* Grass Valley, the furnace had two hearths at different levels connected by a flue 7 ft. 10 in. high. The upper hearth is 6 ft. wide, 39 ft. long, and has six working doors on the side. The lower hearth has the same width but is very much shorter. The furnace has a chimney 25 ft. wide and 28 in. square. The charge is introduced on the upper hearth, and is advanced little by little toward the lower one. It is 9 tons on the upper hearth, and 1 ton on the lower one. Two roasters are all that are required to work the ore. As the ore is finished on the lower hearth a charge of 1 ton is pushed down from the one above, and when this is done a further charge of 1 ton is added on the upper hearth, making 10 tons in the furnace at a time. As the men work eight-hour shifts the capacity of the furnace is 3 tons in twenty-four hours. In front of the furnace there is a cooling floor on which the roasted material is spread out to cool.

At the Union Hill Works there were formerly two furnaces which treated 2 tons in twenty-four hours. One of these is 12 ft. by 14 ft., and has three hearths; the other is 17 ft. by 14 ft., and has but two. The former is only a little smaller than the other, costs no more to build, and does as much work; it is therefore preferred. The ore is rabbled twelve hours on each hearth of the two-hearth furnace, and eight hours in the three-hearth one. It generally takes fire within a few minutes after it falls into the furnace. One cord of wood is used in twenty-four hours in the summer, and one and a half in winter.

Caillot and Company's Chlorination Works at Grass Valley have elliptical furnaces 18 ft. by 15 ft. The outside of the upper part is used as a drying floor. The lower hearth has two doors on each side; the upper one is reached by a platform at the end of the longer diameter, and has one door at one end and two at the chimney. The whole furnace is built of red brick. They treat 2 tons on each hearth, and finish 1 ton in twenty-four hours. The furnace is braced against the brickwork with three 4-in. iron bands; over these, uprights 1 in. by 2 in. are placed, and these surrounded by three bands which are screwed up at the ends. The

* Report of Mining Commissioner, 1871, p. 420.

furnace has lasted five years without any repairs, and would last much longer if it had continuous work; 30 lb. to 40 lb. of salt are added in the roasting, which takes a quarter of a cord of wood to a ton of concentrates. The advantage claimed in using a two-hearth furnace is that it uses less wood than a long one. The shift is twelve hours, and the furnace treats from 2¼ to 2½ tons in twenty-four hours. The charge is drawn every twelve hours; it remains twelve hours in the cubs, and then goes to the cooling floor to be wetted down. There are on the top two charging holes in the direction of the smaller diameter of the furnace. The charge is drawn on to the lower hearth by one opening on the longer axis. There are four openings to discharge into the cubs, one on front of each charging door.

The Petijeans Chlorination Works are not attached to any mill; they are situated two miles below Grass Valley, Nevada County, and purchase the concentrates as they can. The roasting is done in a two-hearth elliptical reverberatory furnace, 18 ft. by 16 ft. The furnace is arched from the four corners, with the dead spaces cut out; the fuel is wood. The lower hearth has two doors on each side. The roasted pyrites are discharged into cubs, and taken from there to the brick cooling floor. The doors of the upper hearth are in the major axis of the furnace at each end. The descending flues for discharging the ore into the hearth below are in the corners. The top of the furnace is used for drying the pyrites; it is held up by rods running through the masonry. Only 1 ton is treated on each hearth at a time; one 1 is drawn every twelve hours. Salt is added on the lower hearth, the temperature of the furnace being allowed to go down for this purpose.

The Pioneer Works, owned by Maltman, three-quarters of a mile from Nevada City, were started in 1858. The furnaces are two long reverberatories, one of which is 70 ft. long on the outside and 12 ft. 4 in. in the clear. The finishing hearths are 16 ft. long, and between this and the roasting hearth there is a drop of 12 in. The other two hearths are 10 ft. wide. The fireplace is 20 in. wide. There are five doors on each side of the drying and roasting hearths, and four on the finishing. The hearth is 2 ft. high to the crown of the arch and 1 ft. at the spring; the arch

generally settles to 20 in. The hearth is flat, with a little spring in the middle, and is made of four layers of brick. The capacity of the furnace is 3 tons in twenty-four hours; it is worked in two shifts; it discharges 1 ton per shift of eight hours. The other furnace is 10 ft. on the outside and 50 ft. long; it has six doors in the long part and three on the finishing; its capacity is 2 tons in twenty-four hours in two shifts. From 30 lb. to 45 lb. of salt is added when the material is dead. The two furnaces treat 5 tons in twenty-four hours, and burn three-quarters of a cord of pine and cedar wood, which cost on an average $4.40 a day. The discharge is made into cubs with a sheet-iron door in front of them. The charges sometimes require 80 lb. of salt.

Merrifield Works, at Nevada City, are situated on the side of the Creek, opposite to the Providence Works. The ore is roasted in a furnace whose first long compartment is 45 ft.; there is then a drop of 1 ft. The finishing chamber is 20 ft. long, making the whole furnace 70 ft. long and 10 ft. wide. There are six doors in the first compartment, and three on the finishing hearth. The furnace takes three charges of 1 ton; 30 lb. to 40 lb. of salt are added, from one-half to four hours before the end of the operation. If the ore contains lime, the salt must be added much longer beforehand; 3 tons are discharged in twenty-four hours, but the furnace contains 5 tons at a time. The amount of wood used for 1 ton of ore is three-quarters of a cord.

At the Providence Works,* Nevada City, Fig. 79, there are two elliptical furnaces with two hearths and a drying floor on top; they are copied from those of Caillot and Co. These, like all the other furnaces, are built of common red brick. The hearths are 15 ft. by 12 ft., and are banded on the outside with bands of 4 in. by 1½ in. flat iron, and after two years' constant use they are in good condition. The ore is first spread out on the top of the furnace to dry. The walls of the furnace extend 10 in. above this level, forming a shallow basin. The ore is passed from one hearth to the next one below through ports 4 in. square, of which there are two in the upper floor, one in the second floor, and four for discharging the ore from the furnace. Each of these ports has a countersunk top, into which an iron plate, with a handle for

* School of Mines Quarterly, vol. v., p. 371.

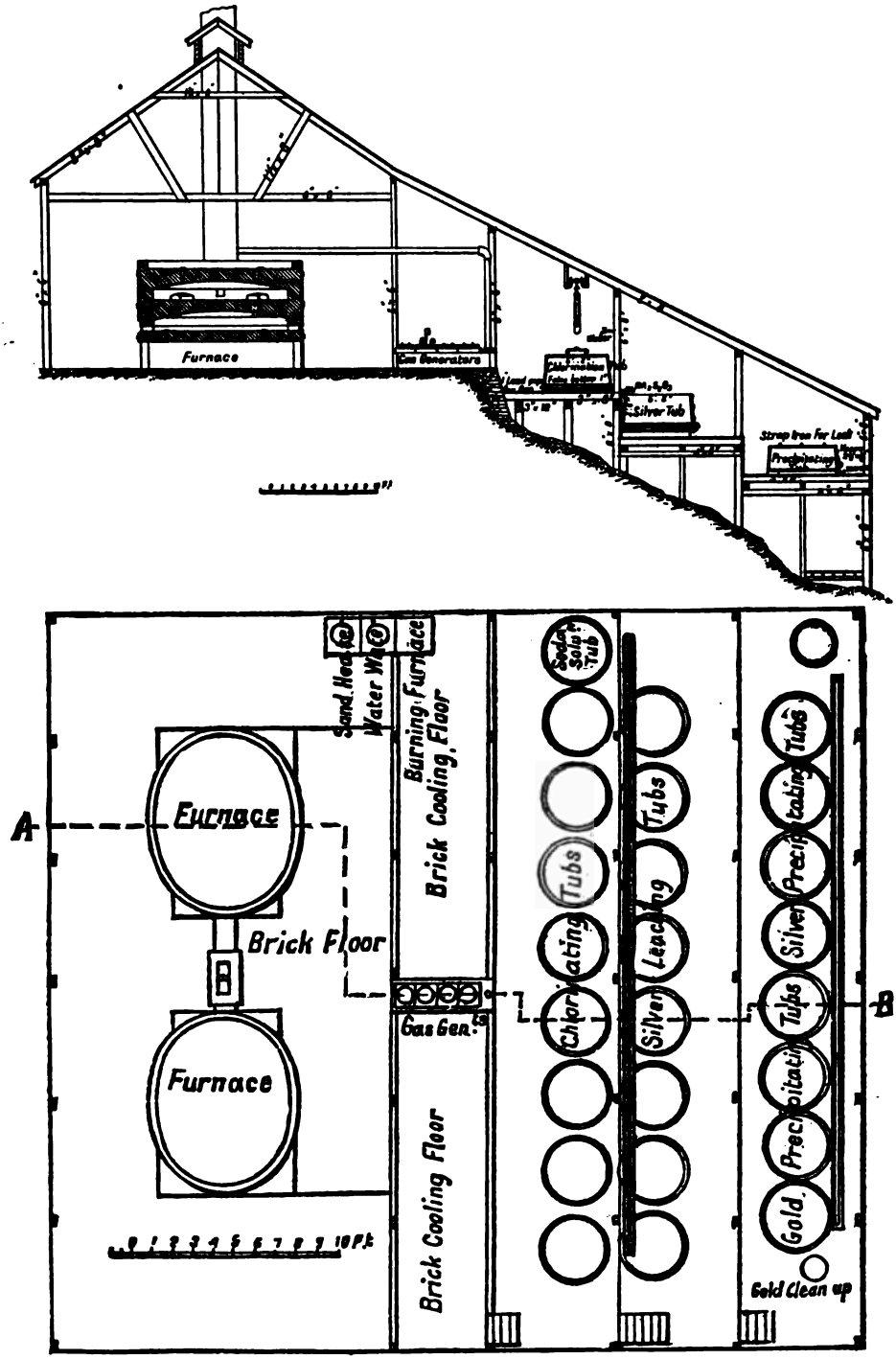


FIG. 79. SECTION AND PLAN OF THE PROVIDENCE WORKS.

its ready removal, is set. The lower hearth is supported on arches, as shown in section through A B. Under these arches the hot ore, as discharged from the furnace, is left to cool. The second hearth is provided with three working doors, and the lower one with four; all of these are arranged with a view to convenience in stirring the ore, and to reaching without difficulty every part of the hearths. The flame spreads well over the lower hearth; the hot gases pass from here to the second hearth, by two flues shown on the plan of the second hearth, and escape from them to the chimney. The walls of the furnace are 13 in. thick, and are held securely in place by iron bands and staves. The brick wall is first encircled by three bands of 4-in. by ½-in. flat iron, each band consisting of four sections which are drawn together by ¾-in. bolts; then thirty-two vertical staves of 1-in. by 3-in. bar iron are set around the wall, as shown, and these are held in place by three outer bands of 4-in. by ⅝-in. iron. Each of these bands likewise consists of four sections, drawn together by 1½-in. bolts which have a right-handed thread cut on one end and a left-handed one on the other, with a square section in the middle. The ends of the outer bands are bent to form rectangular loops which are closed completely by welding, as seen in the drawings. The bolts are screwed into nuts placed in these loops. The working doors are 17 in. wide and 8 in. high at the outside of the wall, but enlarge inward to allow a greater range for the hoe in stirring. In the doorway is set a cast-iron frame ½ in. thick, to which the door is hung. Both flues enter the same stack. In front of the furnace is a space 15 ft. wide, and a brick cooling floor.

There seems to be no advantage in the elliptical form of the furnace. It is much more difficult to build than the ordinary furnace and more difficult to keep in repair. It has been copied from one works to another, simply on the consideration of occupying less space than the Fortschauffelungsofen, but it is not so efficient, and in the end is less economical. The ore, which contains $120 in gold and $12 in silver, is charged on the top of the furnace, all the lumps being carefully broken up, so as to make it as uniform as possible. It is there dried without being stirred. When quite dry the upper hearth being freed by the

discharge of its ore into the lower one, the charging hole is opened, and the ore is pushed down on to the upper hearth through openings made for the purpose. The usual charge is 5 tons. On the upper hearth it is stirred every ten or fifteen minutes, being vigorously rabbled; 2 tons of ore are drawn on to the lower hearth every twelve hours, and the same quantity added from the upper one and as much fresh ore added from the drying floor. After the charge has been four hours on the lower hearth the fire is allowed to go down a little and 1 per cent. of salt is carefully rabbled into the ore. The fire is then raised to the regular temperature, keeping the ore at a bright red heat, and this is kept up until the roasting is completed, or four hours for treating a charge of 1½ tons, and six hours for one of 2 tons. On the lower hearth the ore is constantly stirred up to within half an hour of the time of discharging, when it is raked into a pile and left until it is discharged. One man tends each furnace, dividing his time between the middle and lower hearth. By discharging the ore in one or two hours after adding the salt instead of in from four to six, it was found that more silver was saved but less gold.

The Sierra Buttes Chlorination Works, Sierra County, treat the concentrates from the Frues; the concentrates run from $90 to $300, and average $100. The ore is roasted in the reverberatory furnace, which is divided into two parts by a drop of 8 in.; the part nearest the flue is 36 ft. long, and 12 ft. wide on the outside, and has seven doors on a side; from the drop to the other end is 16 ft., with a fireplace and three working doors; all the doors are on both sides of the furnace. The charge is 1200 lb. to 1500 lb., and there may be as many as five charges in the furnace at one time; the upper end in this way serves as a drying furnace; only one charge is drawn every eight hours. One per cent. of salt is added in the roasting. The smoke from these furnaces is carried from the chimney in a flue 80 ft. long; this flue is 30 in. by 18 in.; a stream of water runs constantly over the top. A water jet from a ¼-in. nozzle, under a pressure of 530 ft., is introduced, to collect the fumes and give a sufficient blast for the draught. This flue enters a tower 4 ft. by 5 ft., and 18 ft. high, in which there are alternating inclined shelves, 6 in. apart, and reaching to 6 in. of

the sides, in which 10 in. of water is constantly flowing in the form of a shower; this runs into a nearly horizontal wooden flue 108 ft. long, which is 24 in. square, with water running over the top; this discharges into a drop-box 12 ft. high, in which there is also a shower of water; the flue then extends 40 ft. to a drop-box 8 ft. high, with another shower, and then discharges into the air. The fumes discharged have hardly any smell.

At the Deloro Mine, Ontario, Canada,* the ore is dried in an inclined revolving cylinder 20 ft. long by 36 in. in diameter at the small end and 48 in. at the large. At the small end there is a conical addition 2 ft. in length, so that the furnace is 22 ft. long. The fire from the fireplace passes through this cylinder to dry the ore. It is dropped mechanically into the roasting furnace, which is a revolving cylinder 30 ft. long by 5 ft. in diameter on the outside lining, with 4½ in. of firebrick, having eight shelves running nearly through from end to end. These shelves are formed of key brick 9 in. long, so that they project 4½ in. from the lining. The fumes are drawn out from the furnace by a fan, which also is used for the draught. The ore runs from the first cylinder through the pipe directly into the second cylinder, 20 ft. long by 48 ft. by 4 ft. in diameter, with a 4½-in. lining, which has six shelves, where the roasting is completed. The ore which feeds furnace No. 1 is heated by the escaping gases of the second cylinder. The two roasting cylinders are jacketted with an air space, and then a covering of mineral wool, and then paper. In these furnaces 10 tons of concentrates have been roasted in twenty-four hours, so that the gold has been extracted up to from 93 to 95 per cent. The roasted ore is left in a pile until the furnace has cooled somewhat, then spread out on the cooling floor in front of the furnace, and when quite cool sprinkled with water and made just damp enough to prevent any dust rising from the ore when it is shovelled. Any lumps that are separated from the mass are ground before being sent to the tubs. In the plan of the works originally designed for Borneo, Fig. 87, it was proposed to use a Brückner's cylinder.

The following Table gives the general dimensions of the fur-

* Trans. Am. Inst. Min. Eng., vol. xi., p. 193.

naces used for working concentrates in some of the principal works:

Name of Works.	Kind of Furnace.	Quantity Worked in Twenty-Four Hours.	Time of Working.	Size of Furnace.	Number of Working Doors.	Number of Hearths.	Number of Furnaces.	Value of Concentrators Treated in Dollars.
Treadwell . .	Spence	8	16	72by11	...	4 { in one plane.	4	40 to 50
Heywood's .	Rever.	3	8	60 ,, 12	9	3 { in one plane.	1	40 to 60
Zeile . . .	,,	3	8	75 ,, 12	8	3 { in one plane.	1	100
Amador . .	,,	4½	8	80 ,, 12	6	3	4	Custom.
Plymouth . .	,,	3	8	50 ,, 12	8	3	1	100 to 200
Sierra Buttes .	,,	1	8	18 ,, 16	10	2	1	103
Petijeans .	,,	1	12	18 ,, 15	4	2	1	Custom.
Caillot's . .	,,	2½	12	50 ,, 12	3	2	1	,,
Maltman's .	,,	2½	8	60 ,, 12	10	3	2	,,
Providence .	,,	4	8	15 ,, 12	4	2	2	140
Merrifield's .	,,	3	12	70 ,, 10	8	6	1	Custom.

Experience has shown that the three-hearth reverberatory roasting furnaces have no advantages over the two-hearth. The top hearth is practically useless, and the furnaces are much more expensive to build. The disadvantages of the two-hearth furnaces are that the top hearth is heated from below, which tends to drive off the sulphur as such, as there is not air enough to burn it, and to sinter the ore. It is almost impossible to do the work properly on both hearths at the same time. On the whole the Fortschauffelung type of reverberatory furnace, such as is used about Grass Valley, answers best. The hearths are from 55 ft. to 65 ft. long, and 10 ft. to 12 ft. wide. They treat about 3 tons a day, the ore remaining in the furnace from forty-eight to seventy-two hours. There is a very great advantage in using the drop furnace either like the one at the Lincoln Works, or the one at Union Hill. Since the ore will be thoroughly scattered by falling from a height it is more likely to be brought into contact with the chloridising gases discharged from the lower hearth. In this respect the furnace at the Lincoln Works seems one of the best used. The Stetefeldt furnace, as at present constructed, with the auxiliary fireplace, is

also in principle a much better furnace, and probably would do the work thoroughly if there were a large supply of ore for constant use. It is such an expensive furnace to build, and the amount to be treated is usually so small, that it is not used.

The fuel used in all these furnaces is spruce or cedar; pine is seldom used, except where no other fuel can be had. The foundation walls of the reverberatory furnaces are generally 2 ft. thick; they are all made of stone up to a little below the fire line. The rest of the furnace is constructed of ordinary red brick, as the heat is not sufficient to make firebricks necessary. The walls of the furnace are generally not more than a brick to a brick and a half thick; they are rarely made two, except the small sides of the drop flues, which are always two bricks thick. The walls of the fireplace are generally one and a half bricks thick. The lower hearth of a two-story furnace, when built with common brick laid up in clay mortar, will last at least four years. The upper hearth will last at least six years with occasional repairs. When the furnace after this period needs to be repaired, 5000 or 6000 bricks will be required for putting it in good condition again. When the ore is discharged from the furnace into the cubs below, the discharge hole in the furnace is covered with an iron plate set in a recess made in the bottom of the hearth, on which a layer of some carbonaceous material is generally put, and then a layer of old ore to prevent caking or attachment of the ore to the iron cover. The spent ore above the carbonaceous material is generally moistened a little to make it pack. The ore is usually drawn from the second hearth upon the one in front very slowly, so that it may be chloridised as far as possible, and from that to the third, and so on, the new charge being introduced on to the upper hearth immediately after the discharge from the lower ones.

As a general rule, all this work is done by the men in charge of the furnace. The assistant is never allowed to discharge the ore from the second to the lowest hearth until the salt has been introduced. The roaster takes charge of the ore on the first hearth, while his assistant pushes it down. It is at a dark red heat and consists of metallic oxides, sulphates of alkaline earths, and a very small percentage of sulphides.

The heat of the furnace, previous to discharging the ore, has been brought up to a bright red. When the ore, on being very suddenly turned over, shows only a few sparks, the salt is introduced. It is not usually put in until after three or four hours' continuous stirring upon the lower hearth in a two-hearth furnace, or in the middle one in a Fortschauffelung. It is always desirable, whether lead is present or not, to add a little salt in the last roasting, as it reduces the amount of chlorine gas consumed, and gives a purer precipitate of gold, but it must be done cautiously and at a low heat. The quantity of salt varies from 5 lb. to 100 lb., depending on the amount of silver, lime, magnesia and lead present. If the quantity of foreign substance is very large, previous to its treatment with the chlorine, the roasted ore should be lixiviated to remove any soluble sulphates and chlorides which have been formed in them, and which might prevent the action of the gas. It is rare, however, that any large amount of such substances is present, and when only a small proportion of impurities are contained in the ore lixiviation is unnecessary, as the water used in dampening the roasted ore appears to attack the alkaline salts sufficiently to permit the gold to be acted on. On an average only about 60 per cent. of the salt that is used is ever decomposed. When the amount of foreign material is large, however, lixiviation must be used. And if the amount of silver contained in the gold is large the lixiviation process is also necessary to separate the silver after the soluble gold chlorides have been washed out.

The roasting must always be commenced at a very low temperature, in order to avoid the formation of salts, which are likely to be reduced in case other metals, especially lead, are present, and also to avoid the fusion or sintering of the ore; and not only must a low heat be maintained during the whole of the first stages, but the whole of the ore must be vigorously stirred during all that time. If, however, a low temperature were maintained, all of the sulphates which are formed would not be decomposed, and there would be danger that all of the sulphur would not be driven off from the ore. If soluble sulphates were allowed to remain in the ore lixiviated, this would precipitate the gold in the tails. The roasting of the ore must therefore be

commenced at a very low temperature and finished at a high one, so that it should be as far as possible "dead" roasted, in order to decompose any sulphates that might be formed. During the roasting air is introduced through the working doors, and the ashpit is kept entirely closed.

On the finishing hearth the ore is raked first from one side to the other, and turned bottom side up with the spadelle, in order to renew the surfaces. It is finally made into piles which are cut down little by little with the same object. When the ore will maintain nearly a vertical face, and shows no bright specks, is inclined to become black, which will generally be in from seven to eight hours, it is ready for leaching.

By the roasting, if carefully conducted, any arsenic or antimony present in the charge is oxidised and volatilised; the sulphides of iron, copper, zinc, and silver are oxidised; and by the addition of salt most of the silver is converted into a chloride, and some copper chloride is formed, while the rest of the base metals are completely oxidised. The gold is left in a free metallic state.

The largest amount of gold has been shown to be in a condition to become soluble, and the least amount of chlorine gas to be consumed to put the gold in that condition, when the ore falling in the furnace has a slight violet colour. When the ore sparkles and the sparks are numerous and bright, a large quantity of salt has to be used.

A ton of roasted ore will occupy generally 24½ cubic feet. This is derived from 2800 lb. of raw sulphurets which occupy about 13⅔ cubic feet per ton. A ton of sulphurets will weigh from 1450 lb. to 1700 lb. after roasting, and will occupy about 17½ cubic feet. It ought not to contain more than 1⅜ per cent. of sulphur.

From the furnace the ore is drawn either into cubs and left in a pile under the furnace to cool or on to a cooling floor, which for a works roasting 3 tons a day must have at least 350 square feet. The surface of this floor is usually paved with brick and inclosed with a rim of ⅛-in. boiler plate 18 in. high. When quite cool the roasted ore is sprinkled with water and thoroughly intermixed. Only just enough water is used to prevent any dust from rising

when the ore is shovelled over; its consistency is such that it barely adheres when squeezed in the hand. The finer the ore the less water is required. The ore is next passed through a six-mesh sieve; all lumps hereby separated from the mass are ground and reroasted. It is necessary that the ore should be entirely cold before being placed in the impregnating vats, as it has been found by experience that warm ore requires a greater quantity of gas, and less gold is likely to be extracted from it.

2. *Impregnation.*—When the ore has become entirely cool, it is dampened with from 4 to 5 per cent. of water, so that it will pack in the hand, but gradually fall to sand when the pressure of the hand is released. It is then either sifted directly into the impregnation vats through a six-to-the-inch sieve, or after sifting is collected in wooden boxes holding about 50 lb. each and emptied gently into the vats. It is even in some cases shovelled in, in which case it is in danger of becoming too closely packed and of preventing the passage of the chlorine gas. The usual form of these tubs, which generally hold between 2 and 4 tons of roasted ore, is shown in Fig. 80. The same vat is used for both impregnation and leaching. They are made of the best sugar pine 2 in. thick; they are 7½ ft. to 9 ft. in diameter, and 3 ft. deep, and generally have a batter towards the inside of 1 in. to the foot. This is done in order to prevent any space forming on the sides when the ore settles, and thus making points of easy passage for the chlorine gas. The vats are usually placed together in a row, each vat being inclined a little to the front so as to drain it completely. In front of the vats is a launder, C, inclined so as to carry off the liquid running through into a reception vat. All of these vats are thoroughly painted inside and out with three coats. The painting material is different in different regions. As all the leaching is done with cold water, it is usually tar, or paraffine, in some cases asphalte varnish; in others a mixture of coal tar and asphalte melted together and put on hot, is used. This is done both inside and out to prevent the action of the chlorine and leakage. The tubs should always be repainted once a year. At the Providence Mine the bottoms of the tubs are made of 3-in. grooved planks, which

are tongued. The staves are made of 2-in. planks with plain joints, but each joint is lapped on the outside with a batten $2\frac{1}{2}$ in. wide by $\frac{3}{4}$ in. thick. The tub is held together by four iron hoops. With this construction there is no danger that the iron hoops will be attacked by the chlorine. The diameter of the tubs at the top is 5 ft. 9 in., at the bottom it is 6 ft. 3 in. They hold $2\frac{1}{2}$ to 3 tons of ore. These vats are very small. They would

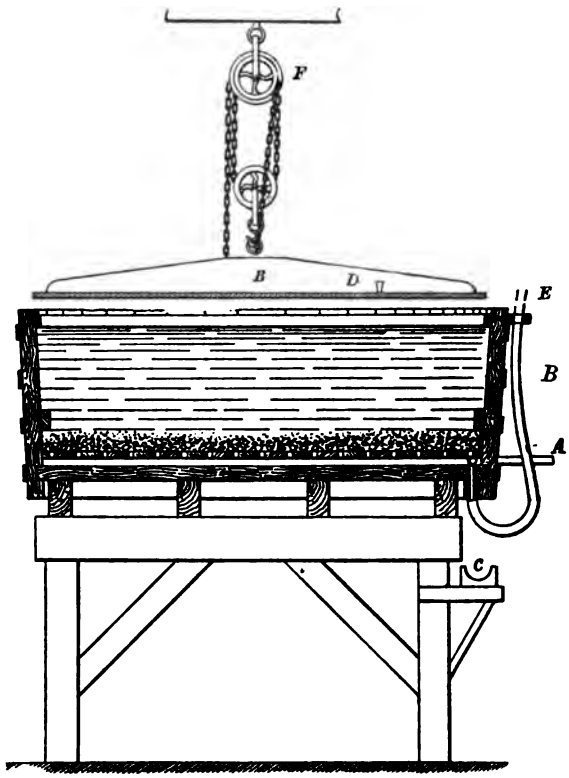


FIG. 80. THE IMPREGNATION AND LEACHING VAT.

work just as well if made ten times this capacity, as is now done in leaching silver ores. The only reason why it is not done is that very few works have such an amount of concentrates at their command as would justify an increase in their size. The filter bottom is made of 1-in. boards placed 3 in. above the bottom of the tub and separated by cleats. The depth of the tub above the false bottom is 3 ft. 6 in. These boards are perforated with $\frac{1}{4}$ in. holes, and are covered with a double

thickness of gunny cloth. A cover is made of 1½ in. stuff tongued and grooved, and reinforced by four wide cleats, which are screwed on. A groove is cut on the circumference of this cover, into which the tub fits loosely; it is suspended by two iron supports on a rolling truck above. These supports are attached to the cleats. The vats contain usually from 2 tons to 4 tons of roasted ore. In most works these vats are placed upon the floor in a line, and on one side of the roasting furnace. In some cases the vats are swung upon gudgeons, as is shown in A, Figs. 81, 84, and 87. In this case the tank is held in place by a hook *b*, and the gold chloride is discharged by a stop-cock K in

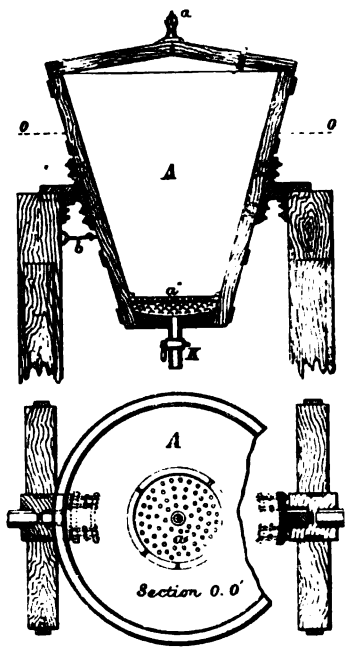


FIG. 81. THE ROTATING LEACHING VAT.

the bottom of the vat. The number of these tanks varies with the quantity of ore to be treated.

At the Treadwell Works in Alaska there are fourteen vats with two smaller ones, making sixteen in all. These are placed eight in a row. Between them, in the centre of the track upon which the ore is brought, is a spout for discharging the spent ore on to wagons below the level of the floor. The vats hold 5 tons and

are filled to 2 in. of the top. They are 9 ft. in diameter and 3 ft. deep on the inside and 3 ft. 8 in. on the outside. They hold a clean 26 in. of ore. They are painted on the inside with paraffine and on the outside with ordinary paint. The filter is made of 3-in. scantling on the bottom, and over this boards filled with $\frac{5}{8}$-in. holes, and on this two layers of sacking, salt sacks being preferred. Around the edge of the tub three to four sacks are carefully placed, and over these slabs about 6 in. apart and 6 in. wide. The interval between them is filled with tails. This makes a floor for the men to shovel on.

At Heywood's Works the leaching tubs, three in number, are 8 ft. 6 in. on the bottom and 7 ft. 6 in. on the top and 44 in deep. They are painted on the inside with paraffine and on the outside with petroleum. The filter is 6 in. deep, made on slats with pieces of quartz the size of a hen's egg, diminishing gradually to grains about the size of wheat, and well graded. It is covered with boards 3 in. wide and $\frac{1}{2}$ in. thick, laid 3 in. apart.

At the Zeile Works vats are 9 ft. 8 in. by 8 ft. 4 in., and 40 in. high. They are painted in different ways, soaked in linseed oil, boiled in wood tar, painted with petroleum on the outside and paraffine paint on the inside. The paraffine paint does not always answer very well, and sometimes chips. One tank made of sugar pine, the staves of which were carefully dried and then boiled in tar, lasted three years. It was then taken to pieces. Two-thirds of the staves were good; the others had to be replaced. It was used again, and at the end of two years was still in good condition. The filter is made of $1\frac{1}{4}$-in. stuff placed on edge over these boards, and then a filter made of graded material up to wheat grain size. This is covered with boards 6 in. apart and the space between them filled with milled sand. A properly made filter lasts two to three years.

At the Amador Works there are three different sizes of tubs, one 8 ft. 6 in. and 42 in. high, two 8 ft., and three 7 ft. 6 in. They are of the same height. These are placed four in a line, one at each end in a double row, with the chlorine generators nearly in the centre of the space between them on the other side of the works. Between them in the floor is a sluice for sluicing out the tails, which are always sluiced away, except when they are

too rich, when they are wheeled out for retreatment. The assay is taken by a probe. The filter is made of 1¼-in. by 1¾-in. scantling placed upon edge. Over these, pieces 1¼-in. by 3-in. or 4 in., with a check of 2 in. between them to keep them apart. On this, quartz the size of a hen's egg, closely packed by hand and then filled with grains down to the size of wheat. Over these, pieces of wood 3 in. by 2½ in., 2 in. to 3 in. apart. The space between is filled with tails. The tubs are painted with paraffine paint, which is found to work best. But for the paint, which wears off the sides of the tank, the filter would last a very long time.

At the Sierra Buttes Works there are three chlorination tubs, 5 ft. in diameter and 8 ft. 2 in. high. They are painted inside and out with tar. The filter is made as usual as to the boards on the bottom, but the filter itself is made of six or seven thicknesses of sacking, which has been found to answer better than a sand filter. Such filters last here from six months to a year.

At Petijeans Works in Grass Valley there are four leaching vats 8 ft. in diameter and 4 ft. high. The filter is made of six series of sifted gravel.

At Caillot's Works the tubs are not painted at all, no paint having been found satisfactory. There are three leaching tubs, 8 ft. by 3 ft. 6 in. The filter is made of gravel, and occupies only a space 8 in. high and 2 ft. long and 2 ft. wide, directly in front of the discharge cock. The gravel is of five sizes. The tub holds 4 tons. There is one tank, 4 ft. by 30 in., which holds 1 ton, which is used for small samples. The leached tails are thrown out of doors and carried away.

At Maltman's Works there are nine tubs 5 ft by 4 ft. The filter is made of boards covered with sacking. Twelve sacks make a layer, and over these 6 in. of washed sand is used. The pulp is sieved in.

At Meredith's Works, Nevada City, there are four leaching vats, which contain 3 tons of sulphurets which have shrunk 25 per cent. in roasting. They are 6 ft. 6 in. by 3 ft. The filter is made by one layer of rocks and no gravel.

The Table on next page gives the dimensions of the vats at some of the principal works.

—	Number of Vats.	Diameter.	Depth Inside.	Capacity in Tons.
Treadwell Mine, Alaska	16	9 ft.	3 ft.	5
Heywood's	3	8 ft. 6 in. by 7 ft. 6 in.	3 ft. 8 in.	4
Zeile	4	9 ft. 8 in. by 8 ft. 4 in.	3 ft. 4 in.	4
Amador	6	8 ft. 6 in.	3 ft. 6 in.	3.5
Plymouth	4	9 ft.	3 ft.	4
Sierra Buttes	3	5 ft.	3 ft. 2 in.	2
Petijeans	4	8 ft.	4 ft.	3
Caillot's	3	8 ft.	3 ft. 6 in.	3
Maltman's	9	5 ft.	4 ft.	2
Providence	8	6 ft.	2 ft. 6 in.	3
Merrifield	4	6 ft. 8 in.	2 ft. 6 in.	3

The false bottom is usually placed at least 1½ in. above the bottom of the tank, in order to allow the exit pipe, which is 1½ in. in diameter, to enter. This false bottom is pierced with ¼-in. holes, 10 in. or 12 in. apart. It is sometimes made by placing pieces of board filled with holes, 2 in. wide and 1 in. thick, from 6 in. to 8 in. apart, over the bottom. These boards rest on cleats several inches apart, which are placed across the bottom of the vat, but do not touch the sides. As, in order to drain completely, the vats when they are set on the floor must be inclined towards the launder in front, the cleats are made thicker on the front side, so that the false bottom is on a level. On this bottom a layer of quartz the size of a hen's egg is spread to the depth of from 3 in. to 4 in. Upon this small pieces half the size are laid from 1½ in. to 2 in. in depth, and so decreasing in size until 6 in. to 8 in. from the bottom sand is placed. The filter is generally 8 in. to 10 in. deep. Over this boards are placed, set in contact to form a surface to shovel upon when the leached ore is to be removed from the tank. These boards are not, however, always used. Sometimes gunny sacking is placed over the filter. When the men are skilful they shovel down on to the filter without disturbing it, but this requires a great deal of practice. This filter remains permanently in the vat until it gets very dirty; it is then removed to be washed and used over again.

When the vats are ready for the charge, fifteen to twenty buckets of dry ore are spread over the bottom, in order to take

up any moisture which may remain there. If it becomes too damp a sufficient quantity of dry ore is added. Wet ore over the false bottom would prevent the easy passage of the gas through it, and cause an increase in the consumption of gas. If it is too wet the whole quantity of gas used for the charge may not rise more than 2 in. above the bottom of the filter. When it is too dry the ore is not properly acted upon. If the bottom of the vat is not dried in this way the lower stratum of ore would take up the moisture and be too damp. The rest of the charge is moistened outside of the vat and treated in the same way. When it is of the proper dampness it is raked together and charged with a shovel on to a brass wire sieve four to eight to the inch, and falls on to the filter. This sieve has wooden sides, and is from 3 in. to 6 in. deep. It either runs upon rollers on two tracks placed at right angles to each other over the top of the vat, or is supported above it by ropes or wires placed at each corner, so as to give it complete freedom of motion. In some cases it has handles on the sides, so as to prevent the labour of reaching over the front in order to get at all parts of the vat; in others it has none, and the men are obliged to lean over the top to make the ore fall in the proper position. By moving the sieve backwards and forwards and from side to side, the roasted ore is distributed as evenly as possible over the bottom, but as a quite even distribution is not possible, it is every few minutes raked evenly over the bottom by the man who throws the ore on the sieve. The object of the screening is to have the charge porous to make it equally damp, uniform in size, and prevent nails, and pieces of iron, which would precipitate the gold, from getting into the vat. In some works the ore is moistened and put into a pile, and is then sieved into wooden boxes, which hold about 50 lb. each, and their contents carefully poured into the vat in such a way as to be sure that no packing takes place. The vat is filled in this way to within from 2 in. to 4 in of the top. It is necessary to leave this amount of space in order to be sure that the whole of the chlorinated ore shall be covered with water when the leaching commences. If this were not so the ore would be unequally moist in different parts, and it would be very difficult to

get all of the soluble gold out of it. When there is about a foot of ore in the tank the chlorine is turned on. It takes one man two hours to charge the vat. He must work rapidly and well not to be caught up with by the gas, for by the time that the vat is completely filled the chlorine is near the top. On the top the ore is made concave, being highest on the sides and lowest in the centre, so as to be slightly funnel-shaped. It is then slightly pressed with the fist around the whole circumference of the vat in order to pack it lightly and prevent the easy passage of the gas there. The gases have always a tendency to follow the sides of a vat if there is an equally easy outlet for them. When the vat is ready, a cover, which is made of flat boards strengthened on top with six wooden braces 2 in. wide and 3 in. high, is placed in position. This is coated on the inside with tar and suspended in many works by means of a chain and a Weston pulley, as is shown at F, Fig. 80; in others there is simply a ring *a*, as in Fig. 81. Where there are a number of vats they are placed in a row, and are served by a Weston pulley running on a rail overhead. There are two ways of closing this vat; either a groove is made in the cover which fits over the edge of the vat, or the cover itself is made to fit the inside of the vat as in Figs. 80 and 84, leaving a space between it and the sides of the tub, but resting upon a projection placed there for the purpose. This joint is usually caulked, first with rags which have been wet, and then is made perfectly tight, either with a mixture of spent ore, bran, and water, or with clay and sand of the proper consistency, or, as is generally the case, with tough dough, as this is the cheapest material that can be used for the purpose. It is always necessary to keep these joints moist during the process, by placing damp rags over them. The gas is introduced in some works by a single pipe A, Fig. 80, K, Fig. 81, and Fig. 84, which enters under the false bottom; in others there is a pipe upon each side, which does not seem to present any special advantage. The pipe is made of lead and is introduced through the sides. It is enlarged from the inside, so as to make it perfectly tight, and then flanged. In some works a lead nipple, treated in the same way, is used to attach a rubber hose, and this seems to be the best disposition. A small hole in the cover is left for the escape of the air, and to

observe by the ammonia test the moment when the chlorine reaches the top. This hole is then closed with a plug D, Fig. 80, *c*, Fig. 81, but the current is kept up until the charge in the gas generator has been completely exhausted. It is generally passed for five to eight, and sometimes twelve hours, the time depending upon the size of the ore. When the gold is very fine it will take less time than when it is coarse. In order to be economical the operation must be conducted quickly. It is necessary, therefore, that the gold should be as fine as possible, or the time taken will be too great.

At the Treadwell Works the tank is filled by shovelling the roasted material on to a sieve made of very heavy wire. This sieve runs on ways, which can be moved to any part of the tub. The ore is sieved in until it is nearly up to the top of the vat. The ways are then moved, and so on, until the tub is nearly full. It is then levelled off with a rake. The vat cover, which is suspended from the roof beams, fits into the tank, which is cut out for the purpose, and is luted on with a paste made of flour and salt and wood ashes.

At Heywood's Works the ore is often charged warm. It is shovelled on to a sieve 3 ft. by 4 ft., which runs on a wooden track, which can be moved over any part of the tank at will. The sieve has a $\frac{1}{4}$-in mesh. The cover is fitted into the top of the tank. The charge is brought up to within 1 in. of the top, and the chlorine turned on. The whole top is then covered carefully with gunny sacking to prevent the unequal distribution of the water when the tank is leached. The cover is $1\frac{1}{2}$ in. thick, and has five cleats 3 in. by $\frac{1}{2}$ in. fastened to it with bolts and nuts, the heads being punched inside and covered with paint. When the cover warps, as it sometimes does, it is weighted down. In the centre of one of these cleats is the hook for raising the lid by the pulley. The top is luted on with a mixture of corn meal and fine tailings, which must be kept moist. It takes the chlorine about three and a half hours to rise to the top when the ore is pure; when it contains lime or manganese, four and a half hours. When the gas rises to 6 in. of the top, ascertained by punching a hole in the pulp, the cover is luted down. Gas is passed for one hour after that. It is then left twenty-four hours to digest.

At the Zeile Works the roasted ore from the furnace is screened on a ⅛-in. mesh screen 50 in. by 36 in. The middle of one end of this screen is hung from the ceiling by a permanently suspended wire and a hook, which is taken down when the screen is. This makes an incline of about 1 ft. in the length of the screen. It is prevented from falling into the tub by two iron rods which fit over the tank and project about 4 in. The roasted concentrates are thrown on to the screen by one man. Another with a hoe moves it up and down on the screen, and with the hoe pulls the screen around where it is necessary. When the tank is two-thirds full it is left until morning. The top of the ore is then turned over with a shovel, and the gas turned on and the tub filled. The cover is then put on and luted. It takes two and half to three hours to gas a tub.

At the Amador Works, when the tub is half full the gas is turned on. It takes five to six hours to gas the tub. They go on filling after the gas is turned on, and when full the outside of the ore is pressed down with the fist for 6 in. or more, and the ore from the centre and the top drawn up with a rake to fill up the depression. The surface of the top is thus made concave and smooth. A single sack is placed under the water hole and its edges covered over with ore to keep it from rising. In this way it is made perfectly flat. The cover has five braces with reinforcements at the edge. It is let into the top of the vat. The rim on the top is brushed out with a whisk broom and the cover let down by a rope. The space between is filled with a paste made of middlings of flour and tails wetted up to a paste. This can be used many times and only requires to be wetted up and fresh material put in to replace the waste. It is put on and taken out with a trowel.

At Merrifield's Works the roasted ore is received on the cooling floor and added in buckets, and when about a carload is in the gas is turned on. At Maltman's Works as soon as about a carload is in they commence to gas. The charger will fill the tub faster than the gas comes up. It takes eight hours to gas, and after the gas is taken off the tub is left for twelve hours, and then leached. The cover is made tight with dough. At Merrifield's Works it takes five hours to fill, but the gas is left on seven

hours after it is filled. Each tub is taken in turn, so that one tub is left in gas four days

At the Sierra Buttes Works the vats are partially filled before turning on the chlorine. The gas remains forty-eight hours in the vat. The cover is put on with dough, and the holes are closed with it instead of corks. On the top gunny sacking is placed to protect the ore against the water pipe. At the Amador Works roasted ore is thrown on to the screen on ways. The screen has ¼-in. holes. A single man does the whole filling.

After the charge is thoroughly impregnated, the chlorine tube is removed, and the gas is allowed to remain in the ore for from

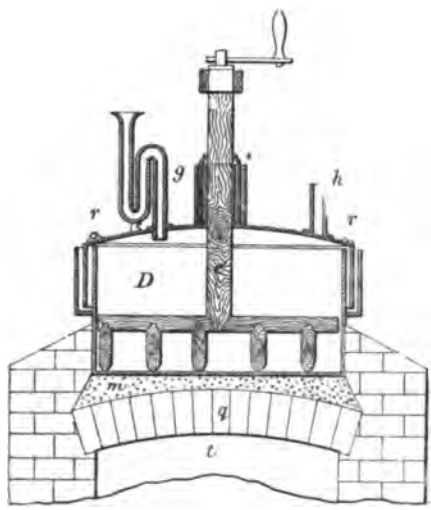


FIG. 82. THE CHLORINE GENERATOR.

twenty-four to forty hours in order to obtain its full action. In the Union Hill Works it was left only twenty-four hours, because they did not have vats enough. This is a very short time, and undoubtedly entails a loss; but they did not seem to think it of very great importance. The best length of time, from the time the gas enters to the discharging of the tub, is from forty-eight to seventy-two hours, depending on the fineness of the ore.

The chlorine necessary for a 3-ton charge of refractory ore is generally made in the covered generator of very heavy sheet of cast lead, or of iron lined with lead, the joints of which have been burned together. They must not be soldered. This is shown at

D, Fig. 82, and is from 20 in. to 24 in. in diameter and 11 in. to 12 in. deep. The cover is made gas-tight by a water joint. It is raised when necessary by means of rings *r* on the side. The bottom is made of lead, which weighs 16 lb. to the square foot. The sides weigh only 8 lb. per square foot. The generator is usually placed on a sand bath *m* standing either on a perforated arch *q* over a fireplace *t* or on a water or steam bath, and is heated up to about 90 deg. Fahr. The water or steam bath is much preferable to the use of the sand bath or hot plate, as it prevents the formation of the hard crusts often formed in the generator from the accidental use of too high a heat. This makes it possible to do away with the stirrer, as is the case in the Merrifield Works, and also makes it possible to remove the exhausted charge very readily by sluicing. Through the centre of the cover a shaft of hard wood *e*, provided with a water joint *s*, and to which a horizontal arm with stirrers is attached, is passed. It has a crank at the top, the object of which is to stir the mixture in the tank after the acid has acted for a certain time. When not in use this agitator is raised 6 in. The gas is prevented from escaping by the hydraulic packing. The charge for 3 tons of ore consists of 20 lb. to 24 lb. of salt, 15 lb. to 20 lb. of binoxide of manganese at 70 per cent., and 30 lb. of sulphuric acid at 66 deg. B. and 18 lb. of water. When necessary, the acid is carried to 45 lb. and the water to 25 lb. The water is not introduced until the gas is wanted. The cover is generally removed to put in the solids and water, but sometimes there is an opening on the top for the introduction of the the charge without lifting the cover, and one on the bottom for removing the spent material. Both of these openings are closed with wooden plugs covered with rags thoroughly saturated with grease to prevent the action of the chlorine. The acid is introduced through a syphon *g*. The bottom of this syphon is often in the shape of a tight-fitting plug, and is large enough when the pipe is removed to admit of the introduction of the manganese. Half a pound of water is added to every pound of acid. This charge will be 3½ in. to 4 in. in depth. The acid is added in pitchers of about half a gallon at a time. In some works the acid is placed in a reservoir whose

stop-cock is directly over the syphon, and the acid is made to fall drop by drop. Such a charge will generally run ten hours.

At Heywood's Works the chlorine is made in lead receivers. The acid is placed in a lead vessel, into which a glass stop-cock is fixed, and flows as it is wanted. The top of the generator is heavily ribbed to make it strong. The water is 7 in. The manganese is stirred as usual from the outside, but the handle has been given up, as it breaks off.

At the Zeile Works the manganese from the chlorine tanks is sluiced out of the mill just like the tails.

At the Amador Works they use two chlorine generators to one washer. The stem of the generator has to be lifted to turn it around. If a crank was used it would break.

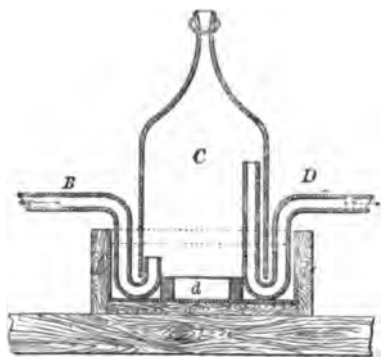


FIG. 83. THE WASH BOTTLE.

At Maltman's Works the chlorinators with the hydraulic seal have been abandoned. In their place are lead ones with only a slight groove on the outside in place of the water seal. This is made tight with dough, which must be kept moist. It is found that this answers much better than a water seal. When it is necessary to move the leaden rake, the dough is taken off, the movement made, and the dough replaced at once, without allowing any inconvenient amount of chlorine to escape. It was found here that the constant breakage from attempting to move the stirrer from the outside was a great inconvenience. As the only wear is made by the stirrer, the bottom up to the point the stirrer reaches is made 1 in. thicker than is usual.

When ready to work, india-rubber tubing, which has been thoroughly greased on the inside, is placed on the exit pipe *h*, which

is also fitted as a plug in the cover, which is $\frac{3}{8}$ in. calibre, and is attached to the entry tube B of the wash bottle C, Fig. 83, and another hose is attached to the exit tube D, which connects it with the leaching vat. The wash bottle generally is a large-sized acid bottle or an old carboy, from which the bottom is removed. It is supported in a wooden box *d* lined with lead and filled with water. The inlet pipes are held in place by a wooden support in the centre of the box *d*. The water in the box stands $\frac{1}{2}$ in. above the inlet pipe. This apparatus is simple, but requires that the pipes should be fixed so that there would be no danger of their turning over, which they might do if not thoroughly supported. Sometimes a special washing apparatus is made. Fig. 84 shows the generator and wash bottle attached to a revolving vat. The sulphuric acid is introduced through the funnel, which is $\frac{3}{8}$ in. calibre, in small quantities at a time. Two of these generators furnish in twelve hours enough gas for one tub, or for 2 tons of ore. It is absolutely necessary that the chlorine gas should be washed entirely free from the hydrochloric acid, since if any sulphides were present they would be attacked, and the sulphuretted hydrogen would precipitate the gold in the impregnation vat. One stick of wood 6 in. to 7 in. in diameter, 4 ft. long, is sufficient to generate all the heat required to make the gas. The gas is given off slowly, and only at about the rate that it can be absorbed in the ore, but it must be given off uniformly. As soon as it commences to decrease in volume fresh acid is added and the stirrer turned. Occasionally the stirrer gets clogged, in which case it must be raised several inches, which can be done without allowing the chlorine gas to escape, as the water joint is quite long. There will generally be no difficulty in breaking up the charge when the stirrer is elevated a little. At Deloro, Canada, the chlorine is made from a charge of 40 lb. to 50 lb. of lime chloride and 50 lb. to 60 lb. of sulphuric acid. It is forced into a cylinder under pressure to about 40 lb. to 50 lb. to the square inch, and falls 25 lb. to 30 lb. when the chlorination has been made as perfect as desirable. In other works the chlorine is made in the same way, but no pressure is used. The operation lasts about two hours. The chlorine generator is generally placed in the centre of the row

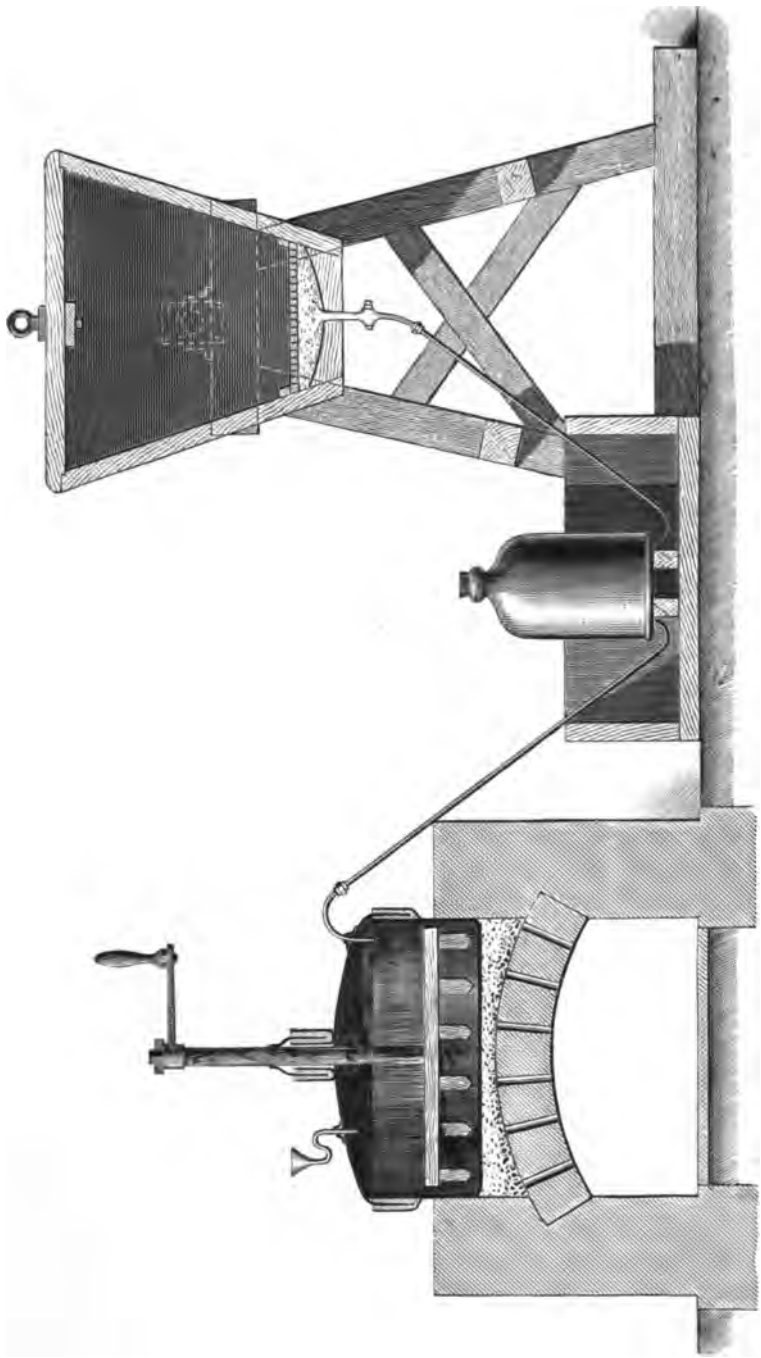


FIG. 84. CHLORINE GENERATOR WASH BOTTLE AND REVOLVING TANK

of vats so as to be as near as possible to them all. If, when the cover of the impregnation vat is lifted previous to the introduction of the water, there is a strong odour of gas, the chlorine has probably attacked the ore thoroughly. In the arrangement as designed in Fig. 87, the generator is placed directly beneath the leaching vat. In the plan, Fig. 88, it is placed in the centre of the row of vats in front of them; in Fig. 79 it is placed a little to one side. There may be one or more of them according to circumstances.

In the Davis process 150 lb. of roasted ore is placed in a cylinder with 125 gallons of water and 50 lb. of lime chloride. The cylinder is then rotated long enough to insure thorough mixture; $3\frac{1}{2}$ gallons of sulphuric acid diluted in the ratio of one to three of water is then forced into the cylinder, which causes the evolution of the chlorine gas. It was thought best to attain a pressure of from 25 lb. to 40 lb. to the square inch in the interior of the cylinder; such a high pressure is not necessary, but only that the pulp should be thoroughly saturated with gas. The operation takes from half an hour to one and a half hours; the gas is then drawn off, which takes half an hour. The cylinder is then discharged into the receiving tank, which has a quartz filter, and the ore washed with 75 gallons of water. All this is pumped into the reservoir, from which it is discharged into the precipitating vats and the spent ore sent to be treated for silver if it contains any. The ore is left in the tubs to digest and chlorinate for two, or preferably for three days. Chlorination should then be complete, which is tested by withdrawing a cork from the cover of the tub and holding an open bottle of ammonium hydrate over the hole; white fumes of ammonium chloride are seen if the chlorine has permeated the ore to the surface and accomplished its work.

3. *Leaching the Impregnated Ore.*—When the charge has remained a sufficiently long time in the vat, which is generally forty-eight hours, the cover is removed. As there is always a great excess of chlorine gas, which completely fills the space between the top of the ore and the cover, if it was lifted at once there would be a considerable escape of gas. In most works this is absorbed by water previous to the lifting of the cover, but as

when the gas is retained in solution it makes it difficult to settle the gold in the precipitation tanks, aspiration is frequently used to draw off the excess of gas and to discharge it into the air outside the building or to collect it in a receiver, from which it is used for a subsequent charge. The introduction of the water for leaching the gold chloride must be done with precaution. In some works, in order to be certain of its equal distribution, the cover is lifted and gunny sacking is placed over the ore previous to the introduction of the water; in others the gunny sacking is placed over the ore before the cover is put on previous to the introduction of the chlorine, and remains on top of the ore while the tub is being gassed. The water is introduced through a hole in the cover, and falls on the gunny sacking and is distributed over the ore. It has been proposed to place permanently under the cover and attached to it a coil of lead pipe filled with small holes. One end of the pipe comes through the cover so that the water is sprinkled all over the ore in fine jets. In this way the most gold chloride is dissolved in the least amount of water; so that the first solution which collects under the false bottom is very rich, while that above is comparatively poor. In any case the vat is filled with water until it is about 3 in. deep over the whole surface of the ore, and the tub is allowed to stand until the water remains at a permanent level and all the gas which is not absorbed by the water has escaped, which is done in from thirty to fifty minutes. The india-rubber tube E, Fig. 80, attached to the exit pipe below the false bottom, is turned down into the launder C, which, like all the launders, is hewn out of a solid stick of timber and is coated with tar. These launders are usually from 4½ in. to 6 in. wide and from 3 in. to 4 in. deep. The solution is allowed to run out very slowly, and water is introduced at the top of the vat as fast as it flows from the bottom. The yellow solution having a strong odour of chlorine, runs through a canvas filter into a vat 18 in. in diameter and 2 ft. deep. In this vat, 2 in. from the bottom, a conical wooden funnel is placed, so that the rubber hose, 1 in. inside diameter, can be slipped over the small end. The object of the canvas bag is to catch any ore or sand escaping from the tanks. Through this tube the solution is carried to the precipitation

tanks. The ore is kept well covered with water, and this is continued until the solution running through shows no trace of gold with an iron sulphate solution. In some works no filter is used.

At the Treadwell Mill, when the tub is quite full, sacking is placed over the ore at the point where the water enters, and the water is allowed to flow through the sacking on to the roasted ore to the depth of 2 in., and to flow in as fast as it flows out, as long as the solution contains any gold. From here it flows into two settling tanks, of which there are eight on a level, 3 ft. below, where it settles. The settlings are collected and added to the roasted ore. From here it flows into six precipitation tanks through three sets of rubber pipes.

At Heywood's Works the cover has three holes: a small one near the edge to try when the chlorine has reached the top and also to let out the gas when it is to be emptied; a large one in the centre to which a pipe was formerly attached to let the excess of chlorine out of the window into the air, instead of into the chlorinating room; and one for the introduction of water. The water is introduced from a 2-in. pipe and runs into a pail which has in the bottom a pipe which fits the water hole in the top of the tank. This hole is 1½ in. in diameter. The water is allowed to run in as long and as fast as it will freely run out of the pail. It takes from one to one and a half hours to fill the tub. As soon as the water is put on, or before the outlet pipe is put on and hung up, the tub is allowed to stand from a half to one hour full of water before being run off, the top being ascertained by a probe introduced through the gas testing hole. It is then discharged into the settling vats.

At the Zeile Works a single bag is put under the water hole before putting on the cover. The cover has seven braces and three holes. When it is so full of water that it will no longer sink in the ore, it is allowed to stand thirty-six hours. The filling is done with a pipe which is put into the hole in the cover. The leaching is done with the water running in as fast as it runs out until there is no indication of gold.

At the Amador Works the tub is left in gas forty-eight hours. When ready for water the outlet pipe is hung up and water allowed to flow in until it is full. It will settle and must be

filled up several times, the number being determined by the quality of the concentrates. As soon as the tank is quite full and all the gas driven out of it, it is ready to leach. The packing is taken out, the outlet pipe turned down into the settling vat, and the cover raised. As the tub is quite full, the water supply must be cut off and the flow allowed to run, so that the water in the tank will not splash when the cover is raised. As soon as it is raised the water is allowed to flow in as fast as it flows out. The pipe is left in the raised cover. One tub being leached will fill three settlers. There are six settling vats. They are 5 ft. 6 in. in diameter and 38 in. deep. When they are full to within 6 in. of the top, 14 lb. to 15 lb. of sulphuric acid is added and stirred in with a wooden paddle and then allowed to settle for four hours. The liquor is then run into the precipitation vats. The gypsum which forms on the sides must be washed with a stream of water to get the gold chloride out of it.

At the Plymouth Works, about 12 ft. behind the leaching vats, which are 9 ft. in diameter and 3 ft. deep, are two gas generators. At right angles to the roasting furnace and 25 ft. from its end are the other tanks, arranged so as to have the bottoms of one set of them just above the tops of the others. At each end of this line are steps to go down to each level. All of these tanks are placed a little above the floor. The settling tanks are 4 ft. 6 in. in diameter and 3 ft. high.

At the Sierra Buttes Works the vat is washed for two hours. The leach water is run into a settling vat 9 ft. by 3 ft. It is settled twenty-four hours. No sulphuric acid is added.

The solution is run into tanks 5 ft. in diameter to be settled, and is drawn off from here to be precipitated. The use of the filter facilitates the operation. In the settling vats, or in the precipitation vats when there are no settling vats, it is the practice in most works to add 32 lb. to 40 lb. of sulphuric acid at 66 deg. B., as it has been found by experience that in such cases the gold precipitates very freely from an acid solution; besides which, the acid throws down the lime which crystallises on the sides of the vat; it is left there until it accumulates to the depth of several inches, as it can always be easily washed. The liquor usually remains in the settling vats

about twenty-four hours. The following analysis of the gold solution taken from a charge of 3 tons of ore yielding $163 to the ton, is given by Mr. Deetken. The sample was taken one minute after the opening of the discharge cock, and was allowed to settle twenty-four hours.

	Grains.
Aluminium sulphate	6.98
Magnesium ,, ...	132.55
Lead ,, ...	4.16
Sodium ,, ...	426.62
Gold chloride	10.44
Iron ,, ...	0.32
Copper ,, ...	8.95
Sodium ,, ...	536.00
Silica	0.24
Free chlorine, by difference	101.02
Total	1227.28

When the ore contains lead the soluble chloride is transformed into sulphate by the sulphates present, and gives the gold solution a milky appearance, but the greater part of this is left on the canvas filter, which must be frequently cleaned. If any of it were to remain in the liquor it would be precipitated as sulphate by the vitriol, and would make the washing of the gold very difficult. In such case the liquor is drawn into a vat where the lead is precipitated by sulphuric acid. It is then decanted and treated as usual. The washing lasts four to six hours, and is continued until the last waters show no trace of gold. The ore is then allowed to drain. Sometimes all the waters from the last part of the washing and from draining, which are poor, are kept separate to be used on the next vat of ore. An assay of the ore in the vat is now made. If it is still rich, which is very rare, it must be again chlorinated. In some works, when it assays $15 or over, the men are forced to do this at their own expense. If the assay shows that the ore has been properly treated, it is shovelled out into a car and carried to the dump heap. This is somewhat laborious. In some works the transfer of the spent ore from one car to another is avoided by having a vat on wheels. It is rolled to the dump heap and there emptied. It requires some care to remove the ore from the top of the filter so as not to injure it. The labour is much reduced when the

vat is suspended, as it can be turned over and the ore in it shovelled out. In other works where the vat is suspended, the spent ore is not shovelled at all, but when the vat is turned down its contents are washed out into a sluice by a hose and disposed of in this way. The spent ore then falls down a slide a^1 into the spent ore car D^1, Fig. 87. The vat when empty is ready for the next charge. Every day one set of vats is charged with gas, a second remains in gas, and a third is filtered and emptied. When there are three or more impregnation vats, one set is filled every fourth day. Filtering and emptying are done every third day. When there are six or more, several are treated at a time.

At Heywood's Works, when the ore is well washed it is discharged. For this purpose, near the vat is a V sluice, on which a splashboard is arranged next to the tub; this board is movable, and answers for several tubs; the tails are sluiced into this, and sluiced out of the works into a stream; the freshets carry them away. The tails run from $1 to $1.50. There are two settling vats 8 ft. by 8 ft., and 6 in. in diameter at the top and bottom; the liquor is allowed to settle until it is quite clear, and then syphoned off.

At the Zeile Works the tails are thrown against a splashboard into a trough, and sluiced out of the works. The wash water flows into two settling vats 8 ft. 6 in. and 8 ft. 8 in. When it becomes tinted it is made to flow into a smaller one; into this 36 lb. of sulphuric acid are added and stirred in and allowed to stand overnight to precipitate the gypsum. This crystallises freely all over the sides and bottom of the tank; it has become 1½ in. in thickness in two and a half years; it must be well washed to get the gold out. Some of the crystals are over an inch long.

At the Sierra Buttes Works the well-washed tails are shovelled into the silver vat, in which the filter is made in the same way as in the others, and are leached with hyposulphite. The hyposulphite tubs are 5 ft. by 3½ ft.; the solution is brought in iron pipes from the storage tank; it is 3 per cent. strong. The leaching takes forty-eight hours; the silver is precipitated with caustic soda and sulphur boiled together; the silver sulphide is roasted in the finishing hearth of the roasting furnace and sold. At Maltman's Works the silver tubs are directly in front of the leaching tubs;

they are leached with a 2 per cent. solution of hyposulphite; they are rectangular, 5 ft. by 4 ft. and 3 ft. deep. The filter is made by covering the false bottom, which is filled with holes, with sacking; no sand is used in them. There are ten of these vats, five on a side. The leaching is done with hyposulphite; the settling tanks are 8 ft. by 3 ft., to which 10 lb. of sulphuric acid are added; they are settled for twelve hours.

At the Providence Works there are eight silver vats for hyposulphite, 6 ft. 6 in. by 3 ft. 6 in., set upon blocks 8 in. high. These vats have strips on the outside to cover the joints.

At Meredith's Works the silver tubs are 3 ft. 3 in. high, and 6 ft. in diameter. The copper and gold are precipitated by iron in tanks 3 ft. 9 in. by 3 ft. 9 in. The tails run $30 to $40. The silver is precipitated by polysulphide of calcium.

4. *Precipitation and Fusion of the Gold.*—The precipitating tanks, Fig. 85, vary from 5½ ft. to 6 ft. 6 in. in diameter at the

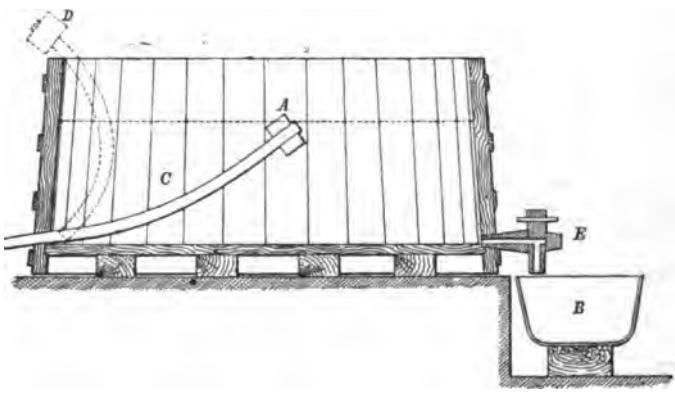


FIG. 85. THE PRECIPITATING VAT.

top, and 6 ft. 6 in. to 7 ft. on the bottom, and are 2 ft. to 3 ft. high. They have no false bottom, and are generally made larger at the bottom than at the top to prevent the adherence of the gold to the sides. They are coated like the solution vats, inside and out, or painted with paraffine paint, or are left without any paint of any kind, and are set entirely level to prevent the loss of gold when decanting the waste liquors. They are made smooth on the inside to facilitate cleaning. The bottom must be made perfectly smooth and is shaped in such a way as to allow of the easiest removal of the precipi-

tated gold. It is made either in Portland cement, or asphalte and cement heated together, and shaped as it cools. The precipitating solution is generally introduced first, so as to neutralise as much as possible the excess of chlorine present in the gold chloride solution, but care must be taken not to introduce too much of it. It is better to add a little afterwards rather than put in too much at the commencement. The gold solution is run into the tanks in such a way as to give it a rotary motion. This facilitates the mixture with the sulphate solution, which is already in the tank, and causes the precipitated gold to collect. In some works the iron sulphate is not introduced until the gold solution is all in the vat. In order to be certain that all the gold is precipitated, it is usual to save some of the strong gold solution which runs out when the first discharge is made, and to add some of the supernatant liquor from the precipitating vat to a small quantity of it. If this precipitates the gold it is certain that there is an excess of precipitating reagent retained in the vat, and that no more needs to be added. It has sometimes happened that after the leach water showed no trace of gold, and the discharge was stopped for a few minutes and again started, there was a precipitate for a few minutes, which would soon stop, and that this might be repeated several times. This is owing to the unequal percolation of the water, and unless guarded against might be a source of considerable loss.

At Heywood's Works the precipitation vats are 6 ft. by 6 ft. 7 in., and 2 ft. 6 in. high; they are painted like the solution tanks; the tub is run nearly full. Two pitchers of sulphuric acid, each holding 26 lb., are added, and the whole well stirred with a stick; seven water buckets, or ordinary pails, of iron sulphate are added, and well stirred in with a wooden stirrer. The liquor is then tested for a change of colour; when there is any change in the colour it is settled for three days with the cover locked on. Eight to nine operations are made in the same vat before discharging.

At the Zeile Works the precipitation is done in eight tubs 6 ft. 6 in. by 6 ft., and 3 ft. 6 in. high; when all the gold is down they are locked and allowed to stand three days, usually in pairs.

At the Amador Works there are six precipitation vats 7 ft. in diameter and 2 ft. 3 in. deep; three to four water pails of iron sulphate are added, a pail at a time, and stirred in; if the liquor turns dark, more must be added; this is done by constant testing. The stirring is done continuously; it is then settled forty-eight hours.

At the Plymouth Works the precipitation vats are 6 ft. 8 in. and 30 in. high.

At Caillot's Works there are four precipitation tubs 8 ft. by 2 ft. 6 in., in which the sulphuric acid is added. There are no settling tanks.

At Maltman's Works there are four precipitation tanks 5 ft. 6 in. by 2 ft. 6 in.

At Merrifield's Works the precipitation tanks are 5 ft. outside by 3 ft. 6 in. inside, and 3 ft. deep.

At Petijean's Works there are two precipitation tanks, but no settling tank.

At the Providence Works the precipitation vats are 6 ft. 6 in. by 2 ft. 6 in., set upon blocks 5 in. high; they are all set so that the bottom of one is on a level with the top of the next in the series.

The precipitating material employed is iron protosulphate, which is prepared at the works, and is generally very acid. It is made by pouring one gallon of carboy sulphuric acid into a tub 4 ft. in diameter, 2 ft. 6 in. high, filled with scrap iron covered with water, so that there is always an excess of iron. The acid is no longer brought to the works in glass, but in iron cylinders 40 in. long and 30 in. in diameter, and is transferred from them for use. One gallon of acid makes a tubful of solution of the requisite strength. This solution is syphoned off into a vat, Fig. 86, 2 ft. 6 in. in diameter at the top and 3 ft. at the bottom and 3 ft. high, and is drawn from thence to be used in the precipitation tanks by a rubber hose A, which is hung up when not in use. When green vitriol, iron sulphate, can be had cheap, a saturated solution of it is made, which is filtered before being used, but generally iron and sulphuric acid are cheaper.

In the Treadwell Mill the iron sulphate is made in five vats between the precipitation vats. At Heywood's Works it is

made in ordinary barrels painted, and any old iron that can be purchased is used.

The gold in the precipitating tanks is generally allowed to settle for twelve to sixteen hours, or at least over night. Sometimes it is necessary to allow it to remain twenty-four to thirty-six hours, for the gold is so often exceedingly fine that a slight purple colour will easily be distinguished for a long time. So long as it can be seen, there is gold in suspension. The time usually depends on the heat of the weather, taking much longer in winter than in summer. When the clear supernatant liquor shows no gold on testing, the upper part of the liquor is removed with a syphon, or by tapping the tank at different levels, as little liquid as possible being left on the precipitate. As both of these methods require watching, the plan generally adopted is to use a

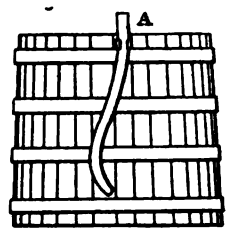


FIG. 86. THE IRON SULPHATE VAT.

flat piece of wood A, attached to a rubber hose C, Fig. 85. This is done in such a way as to keep the mouth of the pipe always under the level of the water, until there would be danger of the gold on the bottom being disturbed, when the width of the float prevents the liquor from being drawn off any further. When not in use the pipe and float are hung up, as shown by the dotted lines D. This method requires no watching. All the liquor flowing off after the precipitation of the gold is collected in large tanks and allowed to settle until place is required for other liquor, when it is drawn off. Very little gold is ever caught in this tank, so that it is only cleaned at long intervals, generally only when the yearly inventory is made.

It is not certain that when the liquor appears clear to the eye there is no fine gold in suspension, so that it is always safe to allow this liquor to flow through a filter of sawdust or sand, as

at E, Fig. 87; it may then be safely allowed to run to waste. If sawdust is used, it is burned after a time and the gold recovered from the ashes; if sand is used, when it is taken up it is chlorinated like the ore. Both of these methods are simple and effective of application.

The precipitated gold, unless the ores are very rich, is allowed to collect for several operations. It is then dipped off from the bottom of the tank with a clean porcelain or enamelled dish, and the bottom and sides of the vat are thoroughly brushed with a small brush and washed with the least possible quantity of water, the final residuum being washed out with the hose from the bottom stop-cock of the vat, settled, and filtered. The precipitation tanks are kept covered and locked except when it is necessary to have them opened to remove the gold or to clean them. When they are not coated with the waterproof paint they absorb a certain amount of the solution, so that when they become old or leaky they are burned to recover the gold from the ashes. They, however, last a very long time.

The precipitate obtained is treated with a mixture of sulphuric acid and salt to free it from iron salts. It is first red and changes afterwards to a brown gold colour. The purified precipitate is placed in a cloth and strained to dry it, and is then squeezed by twisting to make it as dry as possible. When it is moderately pure it will fall after squeezing with a touch of the finger, but when filled with impurities it will not. These cloth filters are always kept under water while they are not in use. When they are worn out they are dried; a little nitre is sprinkled over them; they are then burned and the gold recovered from the ashes. The gold is dried on a pan, but removed before it is completely dry in order to avoid the loss of fine dust of gold, which is very mobile. Sometimes the precaution of using the cloth filter is not considered necessary. The gold is collected on a filter made of punched sheet iron covered with two or three thicknesses of ordinary filter paper and washed with water until it is sweet. It is then dried over a low fire and melted in a cast-iron or graphite retort with salt, borax, and nitre. The gold is generally 800 to 900 fine, sometimes 960, the impurities consisting for the most part of iron

and lead. From $175 to $400, and sometimes more, are taken from the tank at a time. Ninety-five per cent. of the gold is extracted.

At the Treadwell Works the gold which is collected is dried in an iron trough in the circular drum of an ordinary wood stove and is melted in graphite crucibles. The tails run from $1 to $3. If they rise above $4 they are reroasted and retreated. The bullion runs .940 fine.

At the Zeile Works the gold is collected once a month, and washed on an iron funnel covered with a filter paper. It has $\frac{1}{8}$ in. holes 1 in. apart all around. This is dried and fused.

At the Sierra Works the tails run from $3 to $10. Formerly they were always $10. Now they vary between $3 and $4. Some of the old tails are worth working.

This method of precipitation has been almost universally employed up to within a few years. It is difficult, because the fine gold remains so long in suspension, a purple colour being seen in it for several days, and because all the wash water must be kept on account of the small quantity of gold it is liable to contain. The solutions of glue and tannin which have been successfully used in some of the works in Europe, to collect the precipitates of silver, do not seem to have been successfully used upon gold. In order to prevent this loss Mr. H. C. Davis proposes to precipitate the gold from solutions by means of charcoal. According to him it requires 240 grains of charcoal to precipitate 19¼ grains of gold, or about 100 lb. of charcoal to precipitate $2450. To effect the precipitation the gold solution, after it has been allowed to settle, is made to flow through charcoal that has been crushed to pass a screen of twenty-five or fifty meshes. This is placed in barrels having a quartz filter on the bottom; 100 lb. of the crushed charcoal fill the barrel to a height of 28 in. It is desirable that there should be two barrels, the gold solution being introduced into the first at the rate of 20 gals. per hour; the solution passes through it, flowing into the second at the same rate. This is done in order to be certain to catch any gold not precipitated in the first. The liquor from here goes to a collecting tank, perfectly freed from gold, in order to precipitate any other metal that may be mechanically contained in the

solution. As it may contain copper, all of this may be collected as cement copper, and the remaining liquid allowed to flow to waste. After 960 gals. of the gold solution had passed over the charcoal no traces of gold could be detected in the solution passing out of the barrel. Notwithstanding, two barrels are always advisable in case of accident.

At Deloro, Canada, the lime contained in the ore gives with the gold a voluminous precipitate with the iron sulphate, which was principally of calcium sulphate. In order to avoid this the lime was for a time precipitated with sulphuric acid, but this was found too tedious and had to be abandoned. When the precipitation by charcoal was used it was found that the precipitation of the gold was complete.* It was found that the lime, after the liquor passed through the charcoal, undergoes such a chemical change that it is no longer precipitated, either by the iron sulphate or sulphuric acid, although it is by oxalate of ammonia. When a certain volume of the solution has passed through the charcoal it is removed in wooden trays and dried in the air and burned in a muffled furnace. The ash is treated with muriatic acid and washed with water in order to remove any soluble salts. It is then dried and melted in a crucible with fluxes, and usually gives very fine gold. In order to ascertain the exact precipitating value of the charcoal filter at different heights in the barrel, 5400 gallons of one solution were allowed to pass through a bed of charcoal 20 in. thick. The filter was divided into four equal parts of 5 in. each, and these were separately burned. The upper strata gave 2 oz. 95 grs., the second one 2 oz. 99 grs., the third 1 oz. 222 grs., and the last one 1 oz. 62 grs. The use of charcoal has not, however, found much favour on account of the difficulty of recovering the gold from such bulky material.

The precipitation and collection of the gold is a weak and tedious part of the process. It has been proposed to precipitate the gold with sulphurous acid made by heating strong sulphuric acid and charcoal, or sulphur, together in an iron retort. Copper sulphide precipitated for the purpose and arranged on small filters, through which the gold solution is made to slowly filter,

* Trans. Inst. Min. Eng., vol. xi., p. 196.

have also been proposed. The copper sulphide is converted into sulphate; the gold in a metallic state takes its place. When but little copper sulphide remains the gold is removed and cleaned. A little fine sulphur will be contained in it, but this is easily burned off of the gold. The copper is precipitated by H_2S and used over again. It has also been proposed to use H_2S made by melting paraffine and sulphur together and forcing the liquor through a filter press.* There are many advantages connected with the use of filter presses in separating precipitates. They have, however, until recently been little used in this country, though they have been for a long time extensively used abroad.

The ore remaining in the tubs contains the greater part of the silver converted into chloride, which is insoluble in water. It is left in the tubs generally to drain for twenty-four hours. It is then removed to other vats or left in the same vats, and leached with the hyposulphite solution, by which 60 per cent. of the silver contained is extracted. It is better generally to chlorinate the ore first, and leach for silver afterwards. All the work of leaching, impregnation, solution, precipitation, and fusion is done in the daytime; only the roasting is continued during the night.

The Mears process has been used in a number of places in the United States and in Nova Scotia, where it was very successful; it has, however, also been abandoned. It was used at Deloro, Canada, on roasted arseno-pyrite carrying $79 to $90 per ton; 98 to 99 per cent. of the fire assay was extracted in a large way. One of the cylinders used in Nova Scotia contained a charge of 1 ton, and the other carried 3 tons of ore. The charge was chlorinated from two to two and a half hours, lime chloride and sulphuric acid being used in the cylinder. Their experience, however, was that it was much cheaper to make the chlorine in an ordinary generator with manganese and salt and force it into the chlorinator. The pressure of the gas in the cylinder does not seem to hasten the process very much. Its entire virtue consists in the attrition of the particles by the revolution of the cylinder keeping the surfaces of the gold entirely clean to be acted upon by the chlorine. The great objection to the process is that when

* Trans. Am. Inst. Min. Eng., September, 1884.

the lead lining of the cylinder becomes damaged it is extremely difficult to repair it. There is no advantage in this process with coarse gold over the ordinary method for treating it, but it appears that when the gold is coarse all the fine can be readily extracted, and then the coarse gold is in a peculiarly suitable condition for amalgamation, as it is very dry, and is taken up by the quicksilver in less than one-third of the time required before the treatment.

At the Treadwell Mill the solution drawn off of the precipitated gold is run into a tank with an ordinary filter bottom, four thicknesses of gunny sacking and 10 in. of tails, which at the end of the season are taken up and treated as ore.

At Heywood's Works the liquor after the precipitation of the gold is drawn off with a syphon. This liquor is strained in a tank 6 ft. by 5 ft. 6 in. high with a filter made like the others, but covered with several thicknesses of sacking. The outflow from here runs into a tank 6 ft. 5 in. by 2 ft. 6 in. high, and from there to one double that size. These are emptied once in six months and chlorinated.

At the Zeile Works the liquor is allowed to run to waste.

At the Amador Works the liquor is syphoned over the side into a tub 2 ft. by 1 ft., which connects with a sluice which carries the liquor into a tank 12 ft. by 36 in. On the bottom of this tank is a filter made in the ordinary way but with sacking over it. This tub is cleaned once a year. What is collected from it is chlorinated in tubs by itself. The tubs for the sulphate are the same size as the settling vats. There are two of them.

The waste liquor at the Sierra Buttes Works is run into two tanks 16 ft. by 4 ft., and 2 ft. 6 in. deep, and precipitated with old iron. The gold remaining in it and the copper come down. The cement copper is worth from $700 to $800 a ton.

At Petijean Works when the gold is precipitated the liquor is run into two tanks on the outside filled with iron. This precipitates the copper and the gold which has not been precipitated. The cement copper is worth from $40 to $50. It is melted and sold. When the tails are high they are sold to the lead furnaces as a flux.

At Caillot's Works the liquor is drawn off by a lead syphon.

That from the precipitation tanks is run into two vats 12 ft. by 3 ft., and 2 ft. deep, and allowed to settle five or six days, when it is drawn off. These are cleaned every six months. The tails run from $3 to $6.

At Maltman's Works the liquor running off is precipitated with iron. Three of these vats are 5 ft. by 3 ft., and one 8 ft. by 3 ft. The cement copper is worth $800 to $1000.

At the Providence Works the tails run as high as $35, the reason being that the sulphurets contain a considerable quantity of galena, which forms sulphates and chlorides of lead, and prevents the attack of the chlorine. It is now proposed to concentrate the sulphurets and to sell them to the smelters, as is done elsewhere.

5. *Treatment of the Leached Tails for Silver.*—When the ore contains silver nearly the whole of it has been converted into chloride in the roasting furnace. As silver chloride is insoluble it has not been attacked during the process of chlorinating and leaching the gold, and it can be extracted from the ore pulp. The extraction is based on the solution by sodium or calcium hyposulphite of silver chloride, which results in the formation of a soluble double sodium or calcium and silver hyposulphite; the precious metal can be precipitated from this solution as a sulphide by sodium polysulphide, and the simple sodium or calcium hyposulphite left in solution is used over and over again for leaching.

The ore, deprived of its gold, is left to drain for twenty-four hours in the chlorination tubs; then it is shovelled into another set of similar tubs, and pure water is run through the mass until the soluble salts of the base metals are well washed out, which is the case when the water coming from the ore gives but a slight precipitate on being tested with a solution of the polysulphide of lime. The ore is then allowed further time to drain thoroughly, and thereupon is leached with the solution of hyposulphite of calcium.*

For this operation the leaching tubs should be kept full of the solution, and the outflow regulated to suit the capacity of the precipitating tanks, the solution pump, &c. The leaching is

* See chap. x., vol. i.

continued till the solution from the tubs gives but a trace of precipitated sulphide of silver, when a little calcium polysulphide is added to it in a test-glass. This condition is sometimes reached in a few hours, but two or three days are required to leach some silver ores.

The silver solution having been run into tanks, is precipitated with a solution of calcium polysulphide. Great care is exercised in avoiding the use of an excess of the precipitant, for this would subsequently be carried back into the silver leaching tubs together with the solution of calcium hyposulphite, and would there precipitate silver as a sulphide, which would inevitably be lost. It is always safer to leave a little silver in the hyposulphite solution, for there it is not lost, while furnishing sure evidence that no excess of the precipitant has been used. To detect such excess, take in a test-glass some of the clear solution, from which the sulphide of silver has settled, and add to it a little dilute acetate of lead. This reagent will produce a precipitate of sulphide of lead if a trace of calcium polysulphide is present. If an excess is found, more of the silver solution must be run into the precipitating tank.

The heavy sulphide of silver settles in an hour or two, leaving a clear solution of calcium hyposulphite, which is drawn off into a well, and pumped from there up into a distributing tank for subsequent leaching. The sulphide of silver is removed from the precipitating tanks as often as necessary, and is put into a cloth strainer from which the solution drains off. The sulphide, after drying, is roasted in a small reverberatory furnace; the silver is then melted in graphite crucibles, with scrap iron for a flux, and is finally run into bars. At Maltman's Works there are two silver precipitation tanks 8 ft. by 2 ft. 6 in. At the Merrifield Works these are 3 ft. 6 in. inside diameter and 3 ft. deep. The precipitant is polysulphide. The precipitate is roasted in the reverberatory furnace and sold.

The residue or tailings in the silver leaching tubs, after having been deprived of all mineral of economic value, are discarded. Plattner's process and its modifications save from 90 to 94 per cent. of the gold contained in sulphurets, and when the tails contain silver and are leached to obtain it, at least 60 per cent. of

the silver, but the percentage obtained by the other methods is much less. The quantity obtained in each particular case depends on the completeness of the concentration, length of exposure to weathering, and the process by which the sulphurets are worked. The yield by other methods often falls below 50 per cent., though usually it is higher.

Cost of the Vat Process.

The labour account varies according to the various conditions of each works. At Sutter Creek it takes eight men for the whole in twenty-four hours, one chlorinator, one helper, five men at the furnace, and one who wheels the ore. At the Lincoln Mine there are seven men and the superintendent for twenty-four hours. Five of these men are employed at the furnace. The expense of carrying out the process in 1867, on the authority of Kustel, is given below:

	$
Superintendent	6.00
Four roasters at $3.50	14.00
Three cords of wood at $4	12.00
30 lb. of manganese at 6¼ cents	1.87½
40 lb. of salt at ¾ cent.	0.30
70 lb. of sulphuric at 2¼ cents	1.87½
One man at the vats two days	7.00
Sulphate of iron	0.60
Total for 3 tons	$43.65
Or, per ton	14.55

The Table on the next page gives the total expenses of treating 3 tons and the expense per ton in Grass Valley in 1872 in one of the works which is now abandoned. This shows a considerable reduction of cost in five years.

The cost of working by the ordinary process is given variously at the different works. The Table below gives these various costs.

Name of the Mine.	Cost in Dollars.	Name of the Mine.	Cost in Dollars.
Springfield . .	10	North Star . .	20
Keystone . . .	20	Plymouth . .	10
Stewart . . .	20	Empire . .	20
Moore . . .	15	Providence . .	6
Down's . . .	5	Nevada . .	20
Monmouth . .	20	Sierra Buttes .	16.50

	Expense per Three Tons.		Expense per Ton.
	$		$
Expense of Roasting.			
Two shifts on upper furnace at $1.50 per day, Chinese labour	3.00		
Two shifts on lower furnace, at $2.75 per day, white labour	5.50		
One and two-tenths cords of cedar wood at $4 per cord	4.80	14.60	4.87
Salt, 50 lb., at $8 per ton	0.20		
Freight on salt at $12 per ton	0.30		
Wear and tear on working scrapers	0.30		
,, ,, in furnace	0.50		
Chemicals at San Francisco.			
Salt, 20 lb., at $8 per ton	0.08		
Manganese, 15 lb., at $60 per ton.	0.45	1.93	
Oil of vitriol (66 deg.), 35 lb., at 4 cents per pound	1.40		
Expense of Impregnation.			
Freight on salt and manganese, 35 lb., at $12 per ton	0.21		
Freight on acid, 40 lb., $3 per carboy of 165 lb.	0.64		
,, chemicals to works	0.85		
Expense of carboy, per 40 lb. of acid, carboy returned	0.32	4.78	1.95
Labour at vats, $2.50 per day	2.50		
Flour	0.06		
Wear and tear on buckets and hose	0.20		
Precipitation and Melting of Gold.			
Acid for precipitating liquor, 16 lb.	0.64		
Freight and expense of carboy	0.68	1.98	0.66
Old wrought iron	0.06		
Charcoal, crucibles, and fluxes	0.60		
Management		6.00	2.00
Interest on cost of works, $3000		1.20	0.40
Payment towards the sinking fund		2.40	0.80
Fire insurance		1.20	0.40
Full cost		34.09	11.08

These variations depend on local circumstances. The average cost in 1888 was not far from $16. In Grass Valley the works bid against each other, charging $15 to $17, according to the quantity to be treated at any one time, and guarantee a yield of 80 per cent. of the gold and 60 per cent of the silver, so that it is often cheaper to have the work done than to do it.

At the works of the Plymouth Company* they treat 100 tons of concentrates per month of thirty days. The leaching

* Trans. Am. Inst. Min. Eng., May, 1886.

occupies only twenty-four days in the month. The expenses are:

Roasting.

	$	$
Three men at $2.50 a day for 30 days	225.00	
1¾ cords of wood at $4.25	223.13	
54 lb. of salt ¾ cent.	12.15	
		460.28

Chlorine.

60 lb. of manganese per day at $47 per ton ...	33.84	
68 lb. of salt per day at $15	12.24	
120 lb. of acid per day at $60	86.40	
		132.48

Leaching.

40 lb. of sulphuric acid for 24 days ... ... }		
40 lb. of sulphate of iron for 24 days ... ... }	57.60	
One leacher at $5.50 for 30 days	165.00	
Salary of foreman	125.00	
		347.60
		$940.30
Or per ton of concentrates		9.403

In round numbers, and taking a large number of varying conditions into consideration, the cost may be calculated at about $10 per ton.

At the Providence Mine* there are two roasting furnaces of the capacity of 9 tons in twenty-four hours. Each furnace requires one cord of wood in that time. Two Chinamen take care of the furnaces. There is one white man as a foreman, and another Chinaman, but four men are all that are required about the works. The total cost was:

	$
One foreman...	3.00
One white labourer	2.25
Five Chinamen at $1.50	7.50
Two cords wood at $5.00	10.00
29 lb. binoxide manganese at 2¾ cents	.80
260 lb. salt at 1 cent	2.60
216 lb. sulphuric acid at 2 cents	4.32
Lime, sulphur, and calcium hyposulphite	.30
Illuminating	.20
Extras...	1.00
Total	$31.97

This makes the cost of treatment per ton of sulphurets $3.55

* School of Mines Quarterly, vol. v., p. 372.

when the works are run at full capacity. But the ore contains about 7 per cent. of sulphurets, or $4\frac{1}{3}$ tons in the 62 tons milled daily. This quantity of sulphurets does not keep the two furnaces running at full capacity, but both of them are in continual operation. Most of the expenses remain the same whether running at full capacity or not; the actual cost, therefore, figured on a working basis of $4\frac{1}{3}$ tons daily capacity, approximates as follows:

	$
Labour	12.75
Two cords wood at $5.00	10.00
14 lb. binoxide manganese at $2\frac{3}{4}$ cents	.38
126 lb. salt at 1 cent	1.26
104 lb. sulphuric acid at 2 cents	2.08
Lime, sulphur, and calcium hyposulphite	.15
Illuminating	.20
Extras...	.50
Total per day	$27.32
Adding to this the cost for milling per day	57.50
The total outlay per day equals	$84.82

or $1.37 per ton for extracting the gold and silver from the ore. This estimate makes no allowance for the expenses of general supervision, interest on first cost, and gradual deterioration. The conditions of treatment in these works are, however, very special, and can hardly be considered as a basis for the general cost of treatment elsewhere.

The labour will not in general be far from that at the Amador Works, where there is one man brings in wood, who works ten hours, one man at the furnace, who works eight hours, one chlorinator, who works eight hours, and three labourers by day and two by night. The consumption of chemicals per ton of ore is about as at the Merrifield Works.

	lb.
Salt in the furnace	40
,, ,, generator	6
Manganese	6
Sulphuric acid at 66 B	20

It will thus be seen that the cost of roasting is by far the greatest expense in the treatment of the concentrates, it being more than twice that of the chlorination. When a steady supply of ore can be had the cost of treatment will always be less than

when the works are often idle for want of ore. By adopting the use of the Brückner cylinders, or some other mechanical furnace for the finishing roast, it is probable that a considerable expense might be saved. The margin of profit was formerly very large, for at the Lincoln Mine they told me that their usual charge for custom ores was $25 per ton. The difficulty seems to be the frequent enforced idleness of the works from want of ore.

The reason why this process has not been more generally introduced is its inapplicability to all low-grade ores, and especially to to those which have a gangue which may be attacked in the course of the treatment, and the scarcity of ores even in the districts best suited for its application. The works are frequently stopped for want of ore, though the total amount of ore required in twenty-four hours does not generally exceed more than 3 or 4 tons of ore for each works. When the supply of ore is constant, the profit is large, so that it often pays to erect works, when the mines produce sufficient to keep them constantly running. Stopping is bad for the plant as well as the men.

There is a loss by this process, especially of the coarser particles of gold, which have not had time to become attacked. It has therefore been proposed to treat the residue from the tanks in sluices over quicksilver riffles. A considerable quantity of gold might thus be saved, as the gold is bright and would amalgamate easily; but I do not know that this has ever been tried. The general loss in the whole process when the ore contains no silver to be extracted is less than 5 per cent. By using riffles it could probably be reduced to much less. By using leaching after chlorination most of the loss is obviated.

In selecting a site for a vat chlorination works, the fall of ground, supply of water, and the direction of the prevailing wind must be taken into account. Ground must be selected which has a considerable fall so that the ore or tailings may arrive at the top of the roasting furnaces. This fall should allow, at least, of the use of an inclined reverberatory, and if possible of a terraced furnace, or of a mechanical furnace. If all the ground available was necessary to the leaching, elevators might be used for the ore. From the bottom of the furnace there should be a sufficient incline to allow the liquids to flow into the vats where they are

to be used without pumping. The supply of clear water should be at least from 40 to 50 gallons per hour, and there should be several days' supply of it on hand. The water should be carefully tested to ascertain that it will hold the gold in solution, as it has been shown* that waters containing organic matter will not hold the gold in solution. The gases given off from the furnaces and vats are destructive to vegetation and machinery, and are very injurious to health. The works, therefore, must be situated so that the prevailing winds will not carry them in the direction of any mines, mills, or dwellings.

The works are generally arranged in a long rectangle, one side of which is occupied by the roasting furnaces. The impregnation vats are in a straight line with the chlorine generators in the middle or to one side, generally in front of the impregnation vats. The precipitating vats are at one end of this line, so that the furnaces occupy one side of the works and the vats the other.

Fig. 79 shows the plan of the Providence Works, with a sectional elevation on the line A B. As seen, they are built on the side of a hill, and have all the grade necessary for automatic movement. The two two-hearth roasting furnaces with a tun arranged for drying the ore are on the highest level. The four chlorine gas generators are on the same level with the furnace but at right angles to them, and in the centre of the brick cooling floor in front of them. They divide this floor into two, so that there is one floor for each furnace. On the level below are eight chlorination tanks with an overhead railway for the removal of the covers. On one end of the line is the vat for the hyposulphite solution. On the level below, but contiguous to the chlorination tanks, are eight silver-leaching tubs, and on a still lower level eight other vats, the four to the left being used for precipitating the silver and the four to the right for the precipitation of the gold. At one end of the line of silver tubs is the well for the collection of the hyposulphite solution which is stored in the vat N. From here it is pumped back to be used over again. Next to the end of the line of gold tubs is the tank through which the gold solutions run after precipitation to collect the floating gold. One tub is

* Page 67.

gassed each day. They turn out three tons in twenty-four hours, but could turn out 4½ tons in four shifts, but the work is not so well done when done so fast. The concentrates run from $60 to $200, and average about $100. The tails run from $4 to $8.

Fig. 87 shows the plan of another works arranged in steps with the roasting furnace at the highest level. The ore from the furnace is discharged on to the sieve *d*, just above the impregnation vat A, which is suspended on gudgeons. The covers of the vats are raised by the pulley *o*, the rope of which is attached to rings *r*, on the side of the vat. There are six of these vats, three on each side of the chlorine gas generator D, which delivers the gas under the bottom of the tub. In front are the precipitation vats B, two for each set of three impregnation vats; the spent ore from the vat A is slid over a shute a^1 into the spent ore D^1. The supernatant liquor is drawn off the precipitated gold at different levels by the taps *h* and runs into a large vat E, which is filled with sawdust to collect the gold remaining in the liquor, and from here runs to waste.

This design, so far as the chlorine generator and leaching vats, has been of late years much improved by placing the vat on trunnions so that it can be moved as shown in Figs. 81, 84, and 87, but placing the chlorine generator to one side and attaching both to the wash bottle by independent pipes with a nut so that the wash bottle and the pipes may be entirely removed when it is desirable. The vat cover is removable by a Weston pulley block. The vat is placed on skids and the spent ore is easily removed.

Fig. 88 gives the plan of a works designed by O. H. Aaron for treating gold ores containing silver. The whole of the dry part of the process, including the roasting of the ores and fusion of the gold, is done on one side of the works; the leaching process is done on the other side and at a lower level. The ores are first chlorinated in the vats R, and the gold liquor carried to the precipitating vats U at the opposite end of the line. It is then leached for silver in the same vats R, and the silver liquor conveyed by launders to the silver precipitating vats Q. Two tramways serve the tubs, the spent ore being discharged into the tail cars S.

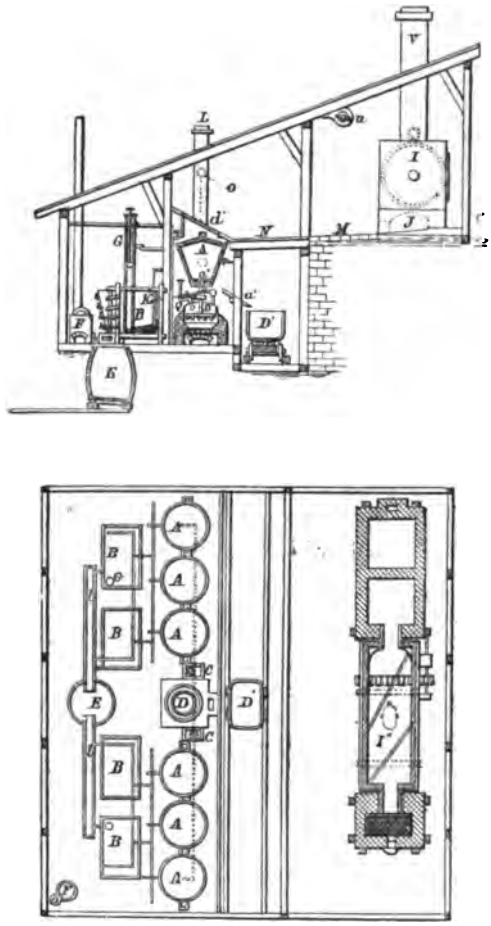


FIG. 87. WORKS WITH BRUCKNER'S CYLINDER AND REVOLVING VAT.

DESCRIPTION.

A. Leaching vat.
a^1. Discharge slide for spent ore.
a^{11}. Filter of the leaching vat.
B. Precipitating tank.
D. Gas generator.
D^1. Spent ore wagon.
E. Vat for spent liquors filled with sawdust.
F. Crucible furnace.
G. Lead hand-pump to raise the second washings into the vat A to be used on fresh ore.
h. Tap holes in the precipitating vats.
I. Brückner furnace.
J. Cub for roasted ore.
M. Brick cooling floor.
N. Wooden floor 34 ft. by 6 ft.
O. Pulley to raise the covers of A.

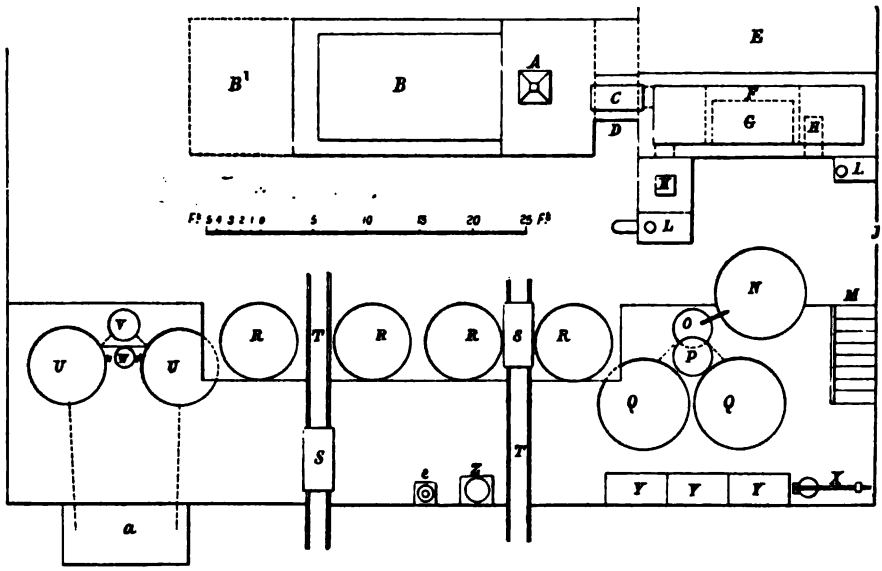


FIG. 88. PLAN OF AARON'S WORKS.

DESCRIPTION.

A. Hopper of the roasting furnace.
B. Two-hearth roasting furnace.
B¹. Same extended to three hearths.
C. Boiler set in flue.
D. Passage under flue.
E. Floor, level with top of drier.
F. Drier.
G. Small furnace in drier.
H. Sheet iron muffle for drying gold
I. Assay furnace.
J. Door to laboratory.
K. Smoke stack.
L. Melting furnace.
M. Steps to lower floor.
N. Tank for hyposulphite.
O. Well and pump.
P. Calcium hyposulphite vat.
Q. Silver precipitating tubs.
R. Leaching vats.
S. Car for removing tailings.
T. Tramways.
U. Gold-precipitating tubs.
V. Iron sulphate vat.
W. Small tub for washings from U.
X. Press for silver precipitate.
Y. Filters for precipitate.
Z. Chlorine generator.
a. Filter tub for waste solution.
e. Wash bottle.

All of these dispositions are excellent. The whole installation, with the exception of the roasting furnace, is very inexpensive, and the wear and tear on the plant is very light. Many other equally good arrangements of the works might be made; but all of these plans are very compact, and give plenty of space, while they are economical in room, and are well adapted to the treatment of such kinds of concentrates as are found in most gold

districts, or of the proper kind of ore. The roasting furnaces adopted are, however, generally the weak side of the process, especially when there is copper in the material treated; but even when there is not, there is loss of gold chloride whenever it is necessary to chlorurise the silver by means of continued heat. The Stetefeldt furnace seems to be the one best adapted to such ores, though there would be undoubtedly a loss even with it; but the instantaneous roasting and chlorurising which the ore receives in that furnace would probably decrease the loss very much, but no experiments in this direction have to my knowledge as yet been made.

The importance of saving sulphurets and working them by Plattner's process is shown in the annual statement of the Sierra Buttes Mine. From 1872 to 1885 the sulphurets were collected by Hendy concentrators, and after exposure to atmospheric oxidation, were worked in the combination Stephenson-Patton pan. In 1885 the company erected chlorination works, at a cost of $5000, by which they have added a yearly profit of $60,000 to the yield of the mine.

The Thies Barrel Chlorination Process.

The difficulties in carrying out the Mears process led Mr. A. Thies, then of the Bunker Hill Mine in Amador City, California, and now of the Phœnix and Haile mines in North Carolina, who for more than four years had been in charge of the Mears process, in which the chlorine produced in a generator is pumped into a barrel under a pressure of 30 lb. to 40 lb. to the square inch, or the same pressure produced by the use of a great excess of chemicals, to endeavour to simplify the process. He found after making a large number of experiments that he could work without pressure pumps just as well as with them, as all pressure necessary for the solution of the gold could be generated within the barrel just as well and much cheaper than by costly machinery, and after making a long series of experiments, leaving out of the Mears process all but the barrel, and having modified that so that there were no joints liable to leakage, in fact so that there was nothing but the bare lead-lined iron cylinder left of it, perfected the process which is now known as the barrel process,

but should be really known as the Thies process, which has been introduced at the Bunker Hill Mine in Amador City, California, and at the Phœnix and Haile mines in North Carolina, where it has been working for several years very successfully.

Through the courtesy of Mr. N. W. Crocker, superintendent of the Bunker Hill Mine, I had an opportunity to study this extremely interesting process at that mine. The details relating to the Haile and Phœnix Mines have been furnished me by Mr. Thies. The Bunker Hill Mine was located in 1852. It is situated 1½ miles north of Amador City. These works, as well as the Phœnix formerly used the Mears process, but were forced to abandon it because the cost of repairs was too great, and it was found to be impossible to keep the chlorine from leaking. There seems to be no doubt that the use of chlorine under very high pressure acts efficiently, and since its abandonment the tails are higher, but the difference is not enough to pay for the extra trouble and expense. This seems, for the most part, to be owing to the use of dry ore and dry gas, no water having been previously used, but the whole difference is not more than $2 per ton, which it seems impracticable to save on account of the difficulty of keeping the chlorine from leaking. All of the Mears process has therefore been abandoned, and as now conducted it does not bear much relation to the original process.

The concentrates at the Bunker Hill Mill are 2 per cent. of the ore. They contain a trace of arsenic, antimony, and lead. Their average value is $60 to the ton. They are roasted with 1 per cent. of salt in a reverberatory furnace with a revolving hearth at the fireplace end. It is 40 ft. long and 12 ft. wide on the outside, with walls 18 in. thick. The stationary hearth is 7 ft. wide, 18 ft. long, and 24 in. high, and has two working doors, one on each side of the furnace. They are 8 in. by 16 in. At the end of this stationary hearth is a drop of 6 in. to 7 in. The ore then falls on to a horizontal revolving hearth 12 ft. in diameter, with a discharge hole in the centre. This part of the furnace is made of an iron shell lined with firebrick. It is 24 in. high to the crown of the roof, and 18 in. to the spring of the

arch. The working door is 3 ft. above the floor of the roasting house. The bridge is 9 in. below the arch, and the grate bars 18 in. below that, so that the fireplace and combustion chamber together are 27 in. deep. The hearth revolves, by means of gear wheels beneath it, at the rate of one turn per minute. The work consists in turning the concentrates over and bringing them from the centre to the circumference, and pushing them from the circumference to the centre. When finished, the charge is shoved from the centre, through the hollow axis of the hearth, into the cub below, and from there the roasted ore is removed by scrapers attached to the bottom of the pan, loaded into iron cars and carried in them to the cooling floor. There are, in the furnace, three batches of 1100 lb. of ore. One is on the end of the hearth nearest the flue drying, the others being roasted. The charges are moved every eight hours. The same charge remains eight hours on the revolving hearth. To ascertain whether the roasting has been sufficiently prolonged to decompose all the sulphates, a small quantity of the hot charge is thrown into water, and a bright iron rod thrust into it. If there is any undecomposed sulphate, the rod will be acted on; if not, the charge is ready for the barrel. The capacity of the furnace is 2 tons in twenty-four hours; 0.25 per cent. of sulphur remains in the ore after roasting. It requires five-eighths of a cord of wood, at $6 per cord, to the ton. At the Haile Mine, in North Carolina, a double reverberatory furnace is used. They have just erected a Spence furnace to roast the concentrates.

The cooling floor is of brick, and is on the same level as the charging hole of the chlorinating barrel. The charge is generally made hot, but not sufficiently so to be burning to the hand. It is weighed and charged in iron wagons. The barrel charging floor is of wood, in the centre of which is a hole 2 ft. in diameter, into which an iron hopper fits. The barrel is on the floor below, and has a manhole 12 in. in diameter, which is turned up underneath this hopper. The barrel is usually made of cast iron, but it is sometimes made of boiler plate. It is lined on the inside with lead usually weighing 10 lb. to 12 lb. to the square foot. All the joints are burned together when the lead is first put in. The lead used is ¼ in. thick, and weighs 12 lb. to the square

foot. After five years of use at the Phœnix Mine, this lining shows no appreciable wear. The barrel is 4 ft. in diameter outside and 40 in. inside, and 6 ft. long outside and 54 in. inside. The heads are made of cast iron with the trunnions cast on them, and are fitted to flanges on the body; these are fitted with tight and loose pulleys. The lining must be well put in, and with great care. It is fastened at once by bolts to the outside, as it has been found very difficult, when repairs are required, to burn lead on which chlorine has acted. There were formerly in the barrel, on the bottom, partitions of lead 2 in. to 3 in. high, which made the charge rise and fall out as the barrel revolves, instead of simply rotating around on the bottom. This has been found to unnecessarily complicate the construction of the barrel, and has recently been abandoned without any perceptible change in the results of the work. When the chlorinating barrel is turned up, 135 to 140 gallons of water are introduced from a wooden barrel at the side of the hopper by a pipe. The exact quantity of water is regulated by marks on the inside of the barrel. As soon as it is empty it is filled again to be ready for the next charge. The quantity of water used must be sufficient to allow the pulp to flow easily, but it must not be too thin; 25 lb. to 30 lb. of dry lime chloride, which is the ordinary bleaching powder of commerce, is now added through the funnel. This amount has been fixed on as the best, as it has been found that with a charge of 40 lb. no better results were obtained. When, as in some cases in North Carolina, the ore contains copper, more bleaching powder has to be used. One ton of roasted sulphurets is then charged and the bunghole closed. The water is added first, because if the dry ore was added and then the water, the barrel would be more than full, as the two would not mix, but the ore falls through the water, so that the barrel is filled up to 8 in. or 9 in. of the manhole. What remains on its sides is then washed off the funnel with a little water. Thirty pounds of sulphuric acid at 66° B. are added after the rest of the charge is made, and the funnel washed free of acid, and removed. An excess of acid must always be used, so as to be sure to convert all the lime into sulphate. In North Carolina the water is added first, then the ore, and then the

acid. The charge which answered best at the Phœnix Mine, where the ore contains copper, is 40 lb. of bleaching powder and 50 lb. of commercial sulphuric acid. At the Haile Mine, where the pyrites is free from copper, the charge is 10 lb. of lime chloride and 15 lb. of acid. Half of each is added at the commencement, and after three hours' rotation the rest. The barrel is rotated until free chlorine is present. When the charge is in, a rubber is placed over the manhole, and an iron plate over this, which is screwed down tight with a long lever as quickly as possible to prevent the escape of any chlorine from the barrel. The barrel is revolved at the rate of twelve revolutions a minute for four hours. Two charges are always made in a day. But at first the whole charge of chemicals was introduced at once. This was found not to work well, and now the charge is divided, half the quantity required to generate the chlorine is introduced at first, and the rest at the end of half the time of rotation, and the process continued. The time of rotation is from four to eight hours, depending on circumstances. When an examination shows free chlorine present the operation is finished. It will not do to trust to pressure alone to determine the presence of chlorine in excess, for other gases are given off as well as chlorine, which would give a false indication if the operation was determined as finished by the pressure alone. A lead valve is arranged in the barrel, so that not only the pressure of an excess of chlorine gas, but its presence and the exact condition of the charge can be ascertained at any time. When examination shows that there is an excess of free chlorine in the barrel, the pulp should be left in contact with it for at least an hour before discharging.

The barrel is surrounded for two-thirds of its height on both sides by a splash-box 18 in. wide, which ends in a trough which can be turned over any one of the tubs. When the charge is finished the barrel is turned up. The chlorinator, wearing a respirator, as there is considerable pressure of gas, loosens the screw of the manhole, so as to raise the rubber a little and allow the extra chlorine to escape for several minutes. He then removes the valve altogether, and with his hands turns the barrel down into the trough. The whole charge runs on to the filter,

the bed of which is 6 in. in thickness. The filter is first flooded from below to the depth of 4 in. or 5 in. with water, and the outlet stopped. This prevents the charge from packing as the ore falls on to it. The barrel is then washed out with a little water, and it is ready for a new charge. The trunnions of this barrel, which was originally made for the Mears process, were made hollow for the purpose of introducing the stationary goose-neck, through which the pressure of the gas in that process, which was from 30 lb. to 40 lb. to the square inch, was made and measured. They were made too large in the first place, and are now very much too large. They have been stopped up with asbestos, as they are no longer used. In all the recently constructed barrels the trunnions have been cast solid. The difficulty with the barrel so constructed was the leakage of the stuffing boxes for the goose-neck, which it was impossible to keep tight. Another and a greater one was the collapsing of the lead lining of the barrel from the exhaust, in the endeavour to collect all of the chlorine gas in the barrel which was not used in the chlorination, so as to use the gas in the next operation. Another was the fact that there was often pressure due to the evolution of other gases than chlorine, which were mistaken for it, and the fact that, from mistakes as to the pressure, the ore supposed to have been fully acted upon by the chlorine had often not been sufficiently treated, so that the tails had to be worked over. The barrel is run by a 30-in. Knight's wheel.

The time necessary to revolve the barrel is not quite fixed, except by the work of the furnace. The total cost of power is $1 in twenty-four hours, or $0.50 per ton. As the furnace is small the quantity to be treated is small. Double the quantity might be treated if there was sufficient pyrites to treat. Two hours with the Bunker Hill concentrates would answer for the chlorination as well as four. The barrel discharges into three filter tanks, which are rectangular, 6 ft. by 8 ft., and 18 in. deep. They are lined with lead, and incline 1 in. toward the drainhole. The filter in California is made in them as usual, of quartz pebbles, gravel, and fine sand. To keep it in place and prevent its surface from becoming uneven, longitudinal slats, 1½ in. deep and 10 in. apart, are used on the top. In North Carolina the bottom of the

tank is first covered with perforated tiles, which are overlaid with gravel and then sand. They have two outlets for discharging into the settling tanks. When the barrel is entirely empty the stopper in the tank is removed, and the charge is allowed to drain, and is then washed with clear water as rapidly as possible, until no gold is left. To do this the barrel charge is allowed to drain until the surface of the pulp is bare. The outlet is then closed, and water is added until it stands 3 in. to 4 in. above the surface of the pulp. This is then drained. The outlet is again closed, and the vat filled full of water, which will be from 9 in. to 10 in. deep. This is drained, and the tails will then generally be clean. They must, however, be examined, and the leaching continued if gold is found. The filtration is done very rapidly, but the time depends for the most part on the fineness of the ore. When the wash water contains chlorine, but no gold, the tails will be clean. From 250 to 300 gallons of wash water are required per ton of ore treated. In six hours from the time the roasted pyrites is charged in the barrel the tails are clean and the filter ready for a new supply of barrel-treated ore.

The settling tanks are round, and are called stock tanks. There are three of these 8 ft. in diameter and 4 ft. deep. They settle in fourteen to sixteen hours, and are drawn into the precipitation tanks, which are on a level below the bottom of the stock tanks. The precipitation tanks are 5 ft. in diameter and 3 ft. deep. In North Carolina they are 6 ft. to 8 ft. in diameter and the same depth. They are quite large enough to hold all the waste liquors of 3 tons of washed ore. There are six of them. The ferrous sulphate is syphoned into them. The solution must have a decidedly acid reaction, in order to be certain that all the lime has been converted into sulphate. Ferrous sulphate gives a bulky precipitate with neutral solutions of calcium chloride. The solution is stirred with a wooden paddle and the ferrous sulphate added in excess. When the gold is all down the ferrous sulphate is turned off, and the whole allowed to settle from forty-eight to seventy-two hours. The supernatant liquor is run off and a fresh charge introduced into the vat. The spent liquor is made to flow on to a sand filter covered with sacking. The sand is taken up once a year and chlorinated by itself. When copper

is present in the ore the weak solution, after the precipitation of the gold, is run over old iron, and then goes to waste. The sulphurets treated assay $55. The tails assay from $3 to $3.50. Occasionally they go as low as $1. The gold is collected, carefully washed to remove the iron salts, and melted as usual. When properly done, the bullion is 990 to 995 fine. Sometimes, however, it may go as low as 975.

It is a matter of surprise, considering the time and pressure of gas used, that this process does not bring the tails down as low as in the Mears process, where the roasted concentrates were moistened with only 15 to 20 gallons of water, whose only object was to prevent the roasted concentrates from dusting. The difficulty seems to be in the roasting. No salt was added in the Mears process, and the temperature of the roasting was much higher at the end. The explanation must be found in the roasting, as the wet gas does not explain the difficulty. No time, quantity of lime, or other change seems to affect the tails. The tails when treated are thrown into a trough in front of the tanks and sluiced out. Only one cord of wood a day is used in roasting. The experience in these works has been that the tanks last best when the wood has been soaked in linseed oil, dried, and painted with tar or three good coats of white lead. The roasted sulphurets are worked up to 92 per cent. of their assay value. When, as they sometimes do, they contain twice their ordinary value, they are worked just as close, the tails in both cases never containing more than $3.50. When they were worked by Plattner's process in the usual way the tails often contained as high as $7 a ton. The cost of this process, at Bunker Hill, per ton of roasted pyrites, is given below:

Roasting.	$	$
Two roasters, at $3.25	3.25	
Five-eighths cord of wood, at $6	3.75	
		7.00
Chlorinating.		
One chlorinator, at $3	1.50	
30 lb. of bleaching powder, at 4 cents	1.20	
36 lb. of salt, at ¾ cent	.15	
20 lb. of 66 sulphuric acid, at 3½ cents	1.26	
Water-power	.50	
General expenses and loss	3.00	
		7.61
Total for one ton of ore roasted and chlorinated		$14.61

The cost of doing this in North Carolina is very much less:

Roasting.	$	$
Four labourers for roasting 2 tons of ore, at $1 ...	2.00	
One cord of wood for roasting 2 tons of ore, at $1.25	.625	
		2.625
Chlorinating.	$	
Two labourers for 4 tons, at $0.90	0.45	
One chlorinator, at $2	0.50	
40 lb. of bleaching powder, at 3 cents	0.30	
60 lb. of sulphuric acid, at 2 cents	0.30	
75 lb. of sulphuric acid for ferrous sulphate, at 2 cents...	0.125	
Repairs, wear and tear	0.20	
Power	0.125	
		2.00
Total for one ton of ore roasted and chlorinated		$4.625

The success of this process is undoubtedly owing to the formation of nascent chlorine in contact with the ore, which is constantly being rubbed bright by the friction of the particles against each other and against the sides in the revolving barrel. In two years' use of the process at the Haile Mine there have been no repairs. Under the conditions of the process it would be impossible for any coating formed on the particles of gold to remain on them, and if the gold is bright it is sure to be attacked, which might not be the case when the ore is treated in vats. The solution and filtration are done so rapidly that there is little chance for the gold dissolved to be precipitated elsewhere than in the precipitation tanks. The advantages of this process are the small amount of space it occupies, the celerity of the operation, the high percentage of yield, the facility of ascertaining the exact condition of the charge at any time, and the very slight wear and tear. The only disadvantages are that a small amount of power is required to be used, which is not necessary in the ordinary Plattner process, and that more than ordinary care and intelligence are required to run it.

So far as I know no other wet processes for the separation of gold from ores are in use in the large works of the United States. The Designole process was tried at the Haile Mill and failed. The Claudet process* has not to my knowledge been used.

* Trans. Am. Inst. Min. Eng., vol. xiv., p. 98.

CHAPTER VII.

PARTING GOLD AND SILVER.

THE UNITED STATES MINTS.

THE mints of the United States Government are situated in the following cities: Philadelphia, Penn.; New Orleans, La.; Denver, Col.; Carson City, Nev.; and San Francisco, Cal. The assay offices are in New York City, N.Y.; Charlotte, N.C.; St. Louis, Mo.; Helena, Mont.; and Boise City, Idaho. All these establishments are rarely in operation continuously, some of them, being small, are often shut down for months at a time.

Previous to the passage of the Coinage Act of 1873, the mint at Philadelphia was styled the U.S. Mint, and was under the personal supervision of the Director of the Mint. The others were styled Branch Mints of the United States, including those of New Orleans, La.; Charlotte, N.C.; and Dahlonega, Ga.; which were all established in 1835, together with the mint at San Francisco, established in 1854, and that at Carson City, Nev., in 1870. The assay offices were established at New York in 1854; at Denver, Col., in 1864; at Boise City, Idaho, in 1872; at Helena, Mont., in 1874, and that at St. Louis in 1881. Coinage was done at the Branch Mint at Charlotte from 1836 until closed by the War of the Rebellion. Its work has since been limited to the business of an assay office, its equipment for coinage having been removed during the war. At the Branch Mint at Dahlonega coinage was done from 1838 to 1861. In 1871 this property was sold by the Government.

By provisions of the Act of 1873 the Bureau of the Mint was established at Washington as a division of the Treasury Department, and the mint at Philadelphia placed under the immediate supervision of a superintendent, the same as in the case of the other mints, which by enactment were specifically

styled the mints at Philadelphia, New Orleans, San Francisco Carson, and Denver, respectively. The functions of the Bureau of the Mint are fully prescribed by the revised statutes and further indicated by special provisions of subsequent appropriation Acts, such, for instance, as relate to the collection of statistics of the production of the precious metals of the United States.

All of these mints and assay offices are regarded by the United States Government as manufacturing establishments, and as far as a vicious political system, which insists on turning out some of the principal officers and men at every change of the politics of the general administration of the Government at Washington admits, are conducted on the same business system as any private manufacturing establishment, the limits of loss within which the work shall be done being regulated by law and strictly adhered to.

The mint at Carson City was established with a view of coining the bullion from the Comstock mines, but for many reasons unconnected with the advantages of position of this mint, as it had but slight advantages over that at San Francisco, although it is only twenty or twenty-five miles from the Comstock mines, their bullion was sent past Carson City to San Francisco. The Carson City Mint is a small one, though it is fully equipped. The mints assay, part, and coin bullion. The assay offices assay bullion of every description, and the New York Assay Office parts it, but it is the only one that does. All the mints except those at Charlotte and Denver make coin.

The coinage of gold, performed exclusively at the mints at Philadelphia and San Francisco, amounted during the year 1885-6 to 5,050,814 pieces, of the value of $34,077,380, against 1,748,158 pieces, valued at $24,861,123.50, in the preceding year.* Of the gold coinage of the year the sum of $27,080,000 was executed at the mint at San Francisco; the rest at Philadelphia. Of the gold coinage $4,871,680 was in double-eagles, $10,428,470

* The Tables and figures relating to the U.S. mints up to p. 697 are taken for the most part from the Reports of the Director of the U.S. Mint for 1885 and 1886.

in eagles, $18,758,145 in half-eagles, $303 in three-dollar pieces, $10215 in quarter-eagles, and $8567 in dollars.

The silver coinage during the year amounted to 31,627,157 pieces, of the value of $30,022,347.95, against 31,699,096 pieces, of the value of $28,848,959.65, in the preceding fiscal year. Of this amount $29,838,905 consisted of silver dollars, $3,052.50 of half-dollars, $3,626.25 of quarter-dollars, and $176,764.20 of dimes. The silver coinage was executed principally at the mints at Philadelphia and New Orleans. The mint at Carson was closed during the entire year, although the coinage of 28,000 silver dollars is credited to that institution. This coinage was really executed prior to June 30, 1885, but not delivered by the coiner to the superintendent until after July 1, 1885, which brought it into this year's statement of coinage.

In addition to the gold and silver coinage, 1,706,651 minor coins were struck, of the value of $17,377.65. 1,696,613 pieces, of the value of $16,966.13, consisted of 1-cent bronze coins; 4519 pieces, of the face value of $135.57, of 3-cent nickel pieces; and 5519 pieces, of the face value of $275.95, of 5-cent nickel pieces. The minor coinage was executed at the mint at Philadelphia.

The coinage executed was as follows:

Description.	Pieces.	Value.
		$
Gold	5,050,814	34,077,380.00
Silver	31,627,157	30,022,347.95
Minor coins	1,706,651	17,377.65
Total	38,384,622	64,117,105.60

In addition to the coinage executed by the mints, the value of the gold and silver bars manufactured during the fiscal year 1886 amounted to:

	$
Gold	19,031,809.21
Silver	8,236,223.77
Total	$27,268,032.98

The production of bars in the previous fiscal year amounted to:

	$
Gold	32,027,463.02
Silver	9,549,313.37
Total	$41,576,776.39

This makes the production less by $14,308,743.39. These gold bars are made of different sizes and of values approximately determined beforehand by the size of the moulds. Their exact value, determined by assay, is stamped on them, with the name of the mint or assay office where they were made. They are everywhere received as coin and in lieu of it for United States taxes and duties, as well as for all other payments, and are much more convenient for large specie transactions than coin, and are besides not subject to loss either from petty thefts, breakage of the package, or abrasion. Many of the depositors of gold dust prefer to receive their gold from the mints in this shape. They are at any time exchangeable for coin at their full value.

The parting processes used in the United States are, the nitric acid process, the sulphuric acid process, and the Gutzkow or silver sulphate method. The first process is used in the mint at Philadelphia only, the sulphuric acid process being used in all the others except in San Francisco, where both processes have been at various times used. The Gutzkow method was formerly used to a small extent in the mint at San Francisco, but has been abandoned there. It has, however, been very largely used in the private refineries, especially in San Francisco, where it was formerly worked on a large scale. During the great tide of prosperity of the Comstock mines it not unfrequently happened that there was more work to be done than the mints at Carson City and San Francisco could perform, so that private refineries sometimes did not only work for private parties, but the surplus work of these mints by this method.

It was found necessary to abandon the use of the nitric acid process altogether on account of the fumes, or to replace it by what is known as the double process, used at the mints of Philadelphia and San Francisco. Recently the sulphuric acid process, where it has been carried on in thickly populated centres with low buildings, has become a matter of considerable annoyance, since the erection of very high buildings in their vicinity frequently puts their upper stories on a level with the top of the chimney of the assay office, which emits sulphuric acid.

The number of ounces of bullion sent to the acid refineries of the coinage mints and the Assay Office at New York to be refined during the year 1885-6 was 7,246,795, from which gold and silver bullion was separated as follows:

	Standard Ounces.	Value.
		$
Gold	1,123,193	20,896,613
Silver...	5,920,573	6,889,393
Total		$27,786,006

This sum was against $37,050,049 in the previous year. In the value of the bullion required to be refined at the mints and assay offices during the year, there was a falling off as compared to the year 1884-5 of $9,264,043.

REFINING BY ACIDS, DURING THE FISCAL YEAR 1886.

Mint or Assay Office.	Gross Ounces.	Gold, Standard Ounces.	Value.	Silver, Standard Ounces.	Value.	Total Value.
			$		$	$
Philadelphia	577,066	135,991	2,530,065	448,337	521,701	3,051,766
San Francisco	1,766,669	367,064	6,829,097	1,536,619	1,788,066	8,617,163
New Orleans	56,350	16,646	309,693	38,953	45,327	355,020
New York .	4,846,710	603,492	11,227,758	3,896,664	4,534,299	15,762,057
Total .	7,246,795	1,123,193	20,896,613	5,920,573	6,889,393	27,786,006

Institutions.	Salaries.	Wages.	Contingent.	Coinage of the Standard Silver Dollar, Act of Feb. 28, 1878 (indefinite).	Total Value.
COIN. MINTS.	$	$	$		$
Philadelphia	41,550.00	293,000.00	100,000.00	...	434,550.00
San Francisco	41,900.00	235,000.00	50,000.00	...	326.900.00
Carson . .	29,550.00	60,000.00	25,000.00	...	114.550.00
New Orleans	31,950.00	74,000.00	35,000.00	...	140,950.00
ASSAY OFFICES.					
Denver . .	10,950.00	14,000.00	6,000.00	...	30,950.00
New York .	39,250.00	25,000.00	10,000.00	...	74,250.00
Helena . .	7,950.00	12,000.00	8,000,00	...	27,950.00
Boisé . .	3,000.00	...	5,000,00	...	8,000.00
Charlotte .	2,750.00	...	2,000,00	...	4,750.00
Saint Louis	3,500.00	...	3,000,00	...	6,500.00
Total .	212,350.00	713,000.00	244,000.00	...	1,169,350.00

EXPENDITURE.

Institutions.	Salaries.	Wages.	Contingent.	Coinage of the Standard Silver Dollar, Act of Feb. 28, 1878 (indefinite).	Total Value.
COIN. MINTS.	$	$	$	$	$
Philadelphia	38,782.89	290,836.29	68,881.16	83,097.99	481,598.33
San Francisco	41,900.00	174,654.59	35,086.70	...	251,641.29
Carson . .	10,345.12	9,085.50	1,990.59	...	21,421.21
New Orleans	31,862.32	73,996.25	32,896.61	36,878.01	175,633.19
ASSAY OFFICES.					
Denver . .	10,435.70	10,735.75	3,257.28	...	24,428.73
New York .	38,828.80	22,909.50	6,617.47	...	68,355.77
Helena . .	7,926.65	11,647.19	7,448.48	...	27,022.32
Boisé . .	3,000.00	...	4,987.17	...	7,987.17
Charlotte .	2,750.00	...	1,125.60	...	3,875.60
Saint Louis	3,500.00	...	1,881.43	...	5,381.43
Total .	189,331.48	593,865.07	164,172.49	119,976.00	1,067,345.04

The appropriations and expenditures for the mints and assay offices of the United States for the year 1886 are given in preceding Tables.

The following Table gives the appropriations for the support of the mints and assay offices for the fiscal year 1887:

Institutions.	Salaries.	Wages of Workmen.	Contingent Expenses.	Total.
	$	$	$	$
Mint at Philadelphia . .	41,550.00	293,000.00	100,000.00	434,550.00
,, San Francisco. .	41,900.00	170,000.00	40,000.00	251,900.00
,, Carson . . .	29,550.00	60,000.00	25,000.00	114,550.00
,, New Orleans . .	31,950.00	74,000.00	35,000.00	140,950.00
,, Denver . . .	10,950.00	14,000.00	6,000.00	30,950.00
Assay office at New York .	39,250.00	25,000.00	10,000.00	74,250.00
,, ,, Helena. .	7,700.00	12,000.00	6,000.00	25,700.00
,, ,, Boisé City .	3,000.00	...	5,000.00	8,000.00
,, ,, Charlotte .	2,750.00	...	2,000.00	4,750.00
,, ,, Saint Louis .	3,500.00	...	3,000.00	6,500.00
Total . . .	212,100.00	648,000.00	232,000.00	1,092,100.00

The following Table gives a detailed statement of expenditures at the United States mints and assay offices during the fiscal year 1886:

Articles.	Ordinary.	Refinery.	Total.
	$	$	$
Acids	2,876.99	41,458.34	44,335.33
Charcoal	5,601.31	1,427.16	7,028.47
Chemicals	10,552.02	1,629.45	12,181.47
Coal	25,307.62	6,743.37	32,050.99
Coke	3,479.35	...	3,479.35
Copper	15,191.82	5,341.31	20,533.13
Crucibles	10,308.49	2,135.65	12,444.14
Dry goods	2,372.75	1,502.58	3,875.33
Gas	8,214.18	1,089.56	9,303.74
Gloves	4,171.81	552.30	4,724.11
Hardware	2,463.44	558.42	3,021.86
Iron and steel	970.03	78.75	1,048.78
Ice	2,715.74	20.94	2,736.68
Loss on sweeps	712.57	700.11	1,412.68
Labour	2,725.36	2,360.25	5,085.52
Lumber	2,437.39	1,194.26	3,631.65
Machinery	2,814.85	1,718.22	4,533.07
Metal work and castings	2,632.85	2,750.98	5,383.83
Mittens and sleeves	3,305.34	2.00	3,307.34
Oil and belting	2,960.97	32.81	2,993.78
Printing	1,101.13	...	1,101.13
Repairs	15,313.78	7,368.21	22,681.99
Salt	13.11	239.72	252.83
Sundries	26,001.21	3,431.27	29,432.48
Wood	11,216.63	685.71	11,902.34
Zinc	...	2,254.98	2,254.98
Salaries	189,331.48	...	189,331.48
Wages of workmen	712,552.82	82,295.62	794,848.44
Total	1,067,365.04	167,461.97	1,234,917.92

	$
Bars manufactured:	
Gold	19,031,809.21
Silver	8,236,223.77
Total	$27,268,032.98
Coinage executed:	
Gold	34,077,380.00
Silver	30,022,347.95
Minor coins	17,377.65
Total	$64,117,105.90

The Table on page 680 gives earnings and expenditures of the United States mints and assay offices for the fiscal year ended June 30, 1886:

EARNINGS AND GAINS.

SOURCES.	Mints.				Assay Offices.						Total.
	Philadelphia.	San Francisco.	New Orleans.	Carson.	New York.	Denver.	Boise.	Charlotte.	Helena.	Saint Louis.	
	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.	dols.
Parting and refining charges	13,067.43	76,355.91	768 09	47.81	92,621.24	..	..	..	..	..	182,855 53
Alloy charge	218.53	2,417.26	8.52	84	962.10	..	..	..	..	..	3,627.65
Charges for assaying, melting, and stamping	..	..	..	..	..	1,475.75	152 47	306.18	1,449.00	167.79	3,551.79
Seignorage on standard silver dollars	3,923,684.87	7,738.77	1,827,209.07	4,618.29	..	..	..	..	..	..	5,763,851 00
,, ,, subsidiary silver	62 38	..	..	..	..	..	..	..	..	..	62.38
,, ,, minor coins	418.61	..	..	..	..	..	..	..	..	..	418.61
Profits on sale of medals and proof coins	2,532 06	..	..	..	..	..	..	..	..	..	2,532.06
Receipts from assays of bullion	268.00	79.00	27 00	3.00	420.00	89 00	122 00	133.25	531 50	114.00	1,738.75
Deposit melting room, grains and sweepings	9 2.93	787.13	448.06	149.53	4,818.38	2,285 97	120.45	199.40	2,068.03	58.00	11,368 46
Surplus bullion returned by operative officers	4 384.42	34,090.32	..	..	16,547.25	..	..	..	..	..	55,022.09
Gain on bullion shipped to the mint for coinage	..	..	..	..	..	2,220.10	601.67	978.14	1,162.54	363.88	5,326 33
Proceeds of of sale old material	163.94	161.00	554.17	..	2.224.10	1,159.00	..	..	80.62	..	5,148.43
Receipts from sale of bye-products	..	..	..	..	17,178.81	..	..	..	..	..	17,178.81
Total	3,946,5.8.47	101,629.49	1,829,610.51	4,819.52	134,282.48	7,180.42	996.57	1,616.97	5,292.29	703.67	6,032,690.39

EXPENDITURES AND LOSSES.

Salaries of officers and clerks	38,782.59	41,900.00	31,862.32	10,345.12	38 828 80	10,435.70	3,000 00	2,750.00	7,926 65	3,500.00	189,331.48
Wages of workmen and adjusters	373,759.73	174,654.29	103,996.06	9,085.50	22.909.50	10,735.75	3,864.75	617.25	11,647.19	1,382.50	712,552.82
Contingent expenses, less amount paid for wastage and loss on sweeps sold	64,269.17	34,729.61	35,146.61	1,635.11	6,617.47	3,257.28	1,122.42	608.35	7,448.48	498.13	156,033.43
Parting and refining expenses, less amount paid for wastage and loss on sweeps sold	5,743.92	18,318.14	1,411.36	584.31	100,497.29	..	..	..	..	..	106,555.02
Wastages of the operative departments	11,713 99	..	7,174.10	..	318.10	..	..	..	..	..	19,206.19
Loss on sweeps of the year	5,760 54	2,564.82	2,406.67	1,640 87	4,764.85	..	..	..	..	..	17,137.25
Expense of distributing silver dollars	90,727.22	11,433.51	38,711.03	26,433.81	..	..	..	..	..	..	167,205 57
,, ,, subsidiary silver coins	454 95	2.90	..	..	..	..	..	..	..	..	457.85
,, ,, minor coins	847.17	..	..	..	..	..	..	..	..	..	847.17
Minor coinage metal wasted	78.96	..	..	..	..	..	..	..	..	..	78.96
Loss on sale of plumbic melts	..	..	..	..	86.08	..	..	..	..	..	86.08
Total	592,838.54	323,603.07	220,708.15	49,724.72	174,022.09	24,428.73	7,987.17	3,975.60	27,022.32	5,381.43	1,429,691.82

The following Table gives the details of the coins authorised by law in the United States:

——	Legal Weight (grains).	Fineness (Thousandths).	Diameter (20ths of an Inch).	Thickness (1000ths of an Inch).
GOLD.				
Double-eagle	516	900	27	77
Eagle	258	900	21	60
Half-eagle	129	900	17	46
Three dollars	54.5	900	16	34
Quarter-eagle	77.4	900	15	34
Dollar (new)	25.8	900	11	18
SILVER.				
Trade dollar	420	900	30	82
Standard dollar	412.5	900	30	80
Half-dollar	192.9	900	24	57
Quarter-dollar	96.45	900	19	45
Twenty cents	77.16	900	17½	47
Dime	38.58	900	14	32
Half-dime	19.2	900	12	23
Three cents	11.52	900	11	18
MINOR.				
Five cents	77.16	75 p. c. copper, 25 p. c. nickel	16	62
Three cents	30	75 p. c. copper, 25 p. c. nickel	14½	34
Two cents	66	95 p. c. copper, 5 p. c. tin & zinc	18	60
One cent	48	95 p. c. copper, 5 p. c. tin & zinc	15	43

The following Table gives the amount of space required to store United States coins:

Description.	Amount.	How put Up.	Space Required.
	$	$	
Gold coin	1,000,000	5000 in 8-oz. duck bags	Nearly 17 cub. ft.
Silver dollars	1,000,000	1000 ,, ,,	250 cub. ft.
Subsidiary silver	1,000,000	1000 ,, ,,	150 ,,

The space occupied by a bag of standard silver dollars, piled snugly in mass, is 12 in. long, 9 in. wide, and 4 in. deep. Small subsidiary silver packs better than dollars. The weight of a thousand dollars in subsidiary silver being 56 oz. less than that of an equal value in standard silver dollars, the spaces occupied by each vary but little from each other.

The following Table gives the highest, lowest, and average value of the United States silver dollar, measured by the gold standard, and the quantity of fine silver purchasable with a United States silver dollar, at the average London price of silver each year since 1873.

Calendar Years.	Gold Value of a Silver Dollar.			Grains of Pure Silver at Average Price Purchaseable with a United States Dollar.*
	Lowest.	Highest.	Average.	
	$	$	$	
1873	0.98,1	1.01,6	1.00,4	369.77
1874	.97,0	1.00,8	.98,8	375.76
1875	.94,1	.97,7	.96,4	385.11
1876	.79,2	.99,1	.89,4	415.27
1877	.90,2	.98,7	.92,9	399 62
1878	.83,9	.93,6	.89,1	416.66
1879	.82,8	.91,1	.86,8	427.70
1880	.87,5	.89,6	.88,5	419.49
1881	.86,2	.89,6	.88	421.87
1882	.84,7	.88,7	.87,8	422.83
1883	.84,7	.86,8	.85,8	432.69
1884	.83,9	.87,1	.86,1	431.18
1885	.79,4	.84,7	.82,3	451.09
1886	.79,7	.71,2	.76,9	482.77
1887	.79,9	.73,3	.75,8	489.98
1888	.75,5	.70,6	.72,7	510.66

The profit accruing to the United States from the treatment is called "seignorage." This is unusually large at the present time on account of the depreciated silver currency which is being coined. The profit is used to replace the losses in gold and silver produced by its treatment. The seignorage on the coinage of silver dollars during the year 1885-6—being the difference between the cost of the bullion and the face value of the pieces coined—amounted to $5,763,851. The seignorage on subsidiary silver coin manufactured at the mint at Philadelphia amounted to $62.38. The total seignorage on silver coin manufactured during the fiscal year was $5,763,913.38.

The following Table gives the ratio of the value of silver to gold each year since 1687.†

* 371.25 grains of pure silver are contained in a silver dollar.

† Report of the Director of the U.S. Mint for 1886, p. 165.

Year.	Ratio.	Year.	Ratio.	Year.	Ratio.	Year.	Ratio.	Year.	Ratio.	Year.	Ratio.
1687	14.94	1721	15.05	1755	14.68	1789	14.75	1823	15.84	1857	15.27
1688	14.94	1722	15.17	1756	14.94	1790	15.04	1824	15.82	1858	15.38
1689	15.02	1723	15.20	1757	14.87	1791	15.05	1825	15.70	1859	15.19
1690	15.02	1724	15.11	1758	14.85	1792	15.17	1826	15.76	1860	15.29
1691	14.98	1725	15.11	1759	14.15	1793	15.00	1827	15.74	1861	15.50
1692	14.92	1726	15.15	1760	14.14	1794	15.37	1828	15.78	1862	15.35
1693	14.83	1727	15.24	1761	14.54	1795	15.55	1829	15.78	1863	15.37
1694	14.87	1728	15.11	1762	15.27	1796	15.65	1830	15.82	1864	15.37
1695	15.02	1729	14.92	1763	14.99	1797	15.41	1831	15.72	1865	15.44
1696	15.00	1730	14.81	1764	14.70	1798	15.59	1832	15.73	1866	15.43
1697	15.20	1731	14.94	1765	14.83	1799	15.74	1833	15.93	1867	15.57
1698	15.07	1732	15.09	1766	14.80	1800	15.68	1834	15.73	1868	15.59
1699	14.94	1733	15.18	1767	14.85	1801	15.46	1835	15.80	1869	15.60
1700	14.81	1734	15.39	1768	14.80	1802	15.26	1836	15.72	1870	15.57
1701	15.07	1735	15.41	1769	14.72	1803	15.41	1837	15.83	1871	15.57
1702	15.52	1736	15.18	1770	14.62	1804	15.41	1838	15.85	1872	15.63
1703	15.17	1737	15.02	1771	14.66	1805	15.79	1839	15.62	1873	15.92
1704	15.22	1738	14.91	1772	14.52	1806	15.52	1840	15.62	1874	16.17
1705	15.11	1739	14.91	1773	14.62	1807	15.43	1841	15.70	1875	16.59
1706	15.27	1740	14.94	1774	14.62	1808	16.08	1842	15.87	1876	17.88
1707	15.44	1741	14.92	1775	14.72	1809	15.96	1843	15.93	1877	17.22
1708	15.41	1742	14.85	1776	14.55	1810	15.77	1844	15.85	1878	17.94
1709	15.31	1743	14.85	1777	14.54	1811	15.53	1845	15.92	1879	18.40
1710	15.22	1744	14.87	1778	14.68	1812	16.11	1846	15.90	1880	18.05
1711	15.29	1745	14.98	1779	14.80	1813	16.25	1847	15.80	1881	18.16
1712	15.31	1746	15.13	1780	14.72	1814	15.04	1848	15.85	1882	18.19
1713	15.24	1747	15.26	1781	14.78	1815	15.26	1849	15.78	1883	18.64
1714	15 13	1748	15.11	1782	14.42	1816	15.28	1850	15.70	1884	18.57
1715	15.11	1749	14.80	1783	14.48	1817	15 11	1851	15.46	1885	19.41
1716	15.09	1750	14.55	1784	14.70	1818	15.35	1852	15.59	1886	20.78
1717	15.13	1751	14.39	1785	14.92	1819	15 33	1853	15.33	1887	21.13
1718	15.11	1752	14.54	1786	14.96	1820	15.62	1854	15.33	1888	21.99
1719	15.09	1753	14.54	1787	14.92	1821	15.95	1855	15.38		
1720	15.04	1754	14.48	1788	14.65	1822	15.80	1856	15.38		

From 1687 to 1832 the ratios are taken from Dr. A. Soetbeer; from 1833 to 1878 from Pixley and Abell's Tables; and from 1878 to 1888 from daily telegrams from London to the Bureau of the Mint.

The balance of profits on the coinage of silver remaining in the possession of the officers in charge of the coinage mints on the 30th June, 1885, was, for the last fiscal year, $725,366.07. Adding to this the profit of the year—$5,763,913.38—makes a total of $6,489,279.45 of silver profits to be accounted for by the mints during the fiscal year. Of this amount the sum of $167,763.42 was paid for expenses in distributing the coin, all of which, except $457.85, was for silver dollars. In addition, the sum of $16,966.87 of the profits was used to reimburse the mints for the losses arising from the wastages of the melters and coiners in the

manufacture of silver dollars, and the loss on the sale of silver in sweeps during the fiscal year.

The sum of $5,751,347.72 was deposited in the treasury of the United States, leaving a balance of $553,201.44 in the coinage mints at the close of the fiscal year, as follows:

	$
Philadelphia...	346,834.99
San Francisco	21,866.96
New Orleans...	184,499.49
Total	$553,201.44

Of this balance the amount at the Philadelphia Mint, $346,834.99, has since been covered into the Treasury, and the balance at New Orleans, $184,499.49 also, leaving the remainder at the mint at San Francisco, which it is expedient to keep at that institution. No silver coinage having been executed at that mint for the last eleven months of the fiscal year 1886, it is necessary to retain a portion of the silver profit fund to pay the cost of distributing the coin still on hand.

The reports of the Director of the Mint, as well as the accounts of the coinage mints on file in the office of the Register of the Treasury, show that the seignorage on silver on July 1, 1878, the beginning of the fiscal year following the passage of the "Act authorising the coinage of the standard silver dollar," amounted to $424,725.47. The seignorage since that date up to the close of the fiscal year ended June 30, 1889, has amounted to $56,939,730.87. To this is to be added the balance on hand at the coinage mints July 1, 1878, the beginning of the fiscal year following the passage of the act authorising the issue of silver dollars. This balance was $424,725.47. There is also to be added $9,237.54, refunded by Adams Express Company for overcharges in their bills for shipping silver dollars, and $4,560.30, consisting of surplus silver bullion and profits arising from the adjustment of silver values, which sum, while not strictly seignorage on silver coinage, was carried to the credit of the silver profit fund. These items make a total of silver profits to be accounted for of $57,378,254.18.

Of this amount the sum of $781,898.88 has been paid for expenses connected with distributing silver coins. The sum of

$246,617.73 has been paid for wastage in connection with the silver dollar coinage and for losses on sale of sweeps attending that coinage, thus leaving the net profit for the eight years, including the balance in the mints on July 1, 1878, on the manufacture of silver coins, $56,349,737.51. Of this amount the sum of $56,274,489.27 has been covered into the Treasury of the United States.

A statement showing in detail the profits on the coinage of silver during the fiscal year, and the disposition of the same, is given on p. 680.

The standard of gold and silver is 900 parts of pure metal and 100 of copper alloy in 1000 parts of coin. Fineness expresses the quantity of pure metal in 1000 parts. 23.22 grains of pure gold is worth $1.00; hence the value of an ounce is $20.67. The value of a grain is $0.043. 371.25 grains of pure silver is worth $1.00; hence the value of an ounce is $1.29+. Standard gold is worth $18.60+ per ounce. Standard silver is worth $1.16 $\frac{4}{11}$ per ounce. Silver coin of less value than $1 is issued at the rate of 385.8 grains to the dollar.

The toleration allowed by law from the standard is for gold 0.015 either way, or that the gold may contain 0.8985 to 0.9015 of gold. Of silver the toleration is 0.003 either way, the coin varying from 0.897 to 0.903. In actual working, however, the Mint regulations are not so lenient, the gold ingots not being allowed to vary more than 0.0005 and the silver 0.0015. The gold ingots, therefore, actually vary only 0.8995 to 0.9005, and the silver 0.8985 to 0.9015.

In adjusting the blanks for coin, before striking them the law allows a deviation of $\frac{1}{2}$ a grain from the legal weight in the eagle and double-eagle, and $\frac{1}{4}$ of a grain in the case of other gold coin. It may happen that a coin is struck $\frac{1}{2}$ a grain light, in which case the wear necessary to make it uncurrent would be only 2.08 grains. In the year 1873 the light coins accumulated since 1834 were recoined. The loss by abrasion on the 16 millions of dollars of light coin, comprising a few double eagles but consisting principally of eagles, half and quarter eagles, and dollars, was .00514, or a little over $\frac{1}{2}$ per cent. for a wear of twenty-five years.

LEGAL WEIGHT, LIMITS OF TOLERANCE AND ABRASION, LEAST CURRENT WEIGHT, &c., OF UNITED STATES GOLD AND SILVER COINS.

Denomination of Coins.	Standard or Legal Weight.	Tolerance allowed by Law on Individual Pieces.	Minimum Legal Weight of New Coins (less Tolerance).	Half per Cent. Abrasion for Twenty Years' Wear.	Least Current Weight after Twenty Years' Wear (Standard Weight less Legal Limit of Abrasion).	Annual Abrasion within Legal Limit.	Actual Coefficient of Annual Abrasion from Experiments.
Gold.	grains.	grains.	grains.	grains.	grains.	grain.	grain.
Double-eagle .	516.0	.50	515.50	2.58	513.42	.129	.0860
Eagle . .	258.0	.50	257.50	1.29	256.71	.0645	.0430
Half-eagle .	129.0	.25	128.75	.645	128.355	.0322	.0215
Three-dollar .	77.4	.25	77.15	.387	77.013	.0193	.0129
Quarter-eagle .	64 5	.25	64.25	.322	64.178	.0161	.0107
Dollar . .	25.8	.25	25.55	.129	25.671*	.00645	.0103
Silver.							
Dollar . .	412.5	1.5	411.0				
Half-dollar .	192.9	1.5	191.40	...	...	...	.214
Quarter-dollar	96.45	1.5	94.95	...	...	...	.137
Dime . .	38.58	1.5	37.08	...	...	...	.077†

* The limit of tolerance of the gold dollar being one-quarter of a grain (nearly double the limit of abrasion), the gold dollar will continue current until reduced in weight below 25.55 grains.

† Estimated at one-fifth of 1 per cent. a year.

The material which comes into the mints and assay offices of the United States consists of gold and silver bricks, coins, plate, old jewellery, amalgam, and, in fact, anything containing gold or silver, or both. Formerly a great deal of Dorè bullion was received from the Nevada mines, most of which contains as high as 5 or 6 per cent. of gold. Occasionally material containing as high as 15 to 65 per cent. of lead is received. In some cases such bullion as this is refused by the United States, and is taken by private refineries. All this material is only taken on deposit, and no matter of what kind it may be, each deposit is weighed in its crude state as it comes in, is marked with a number, melted, and cast into "slipper moulds," making "shoe bars," as they are called, without reference to their contents. Each lot is melted up by itself and is then assayed, its value affixed to it, after which it is turned over to the parting house.

When the deposit is made the depositor receives a receipt stating the weight of the metal in troy ounces and decimals of an ounce. No value is attached to the weights given on the receipt. This value is subsequently determined by assay. On this assay the depositor receives a certificate which is salable, or if there is a surplus of coin, he is given its value in coin, a certain amount being deducted for the expense of working the bullion. The bullion so received is placed in a wooden box with a label stating its history, and a number is chalked on the lid. After the melting this number is stamped on the bullion produced. The only object of the preliminary melting is to make the metal homogeneous, so that an assay may be made of it. After melting the weight is again taken so as to ascertain the loss in melting.

The moulds used for deposit melting are rectangular, cut off at the corners of each end, having generally two separations which reach about half the thickness of the moulds, so that the ingot can be readily broken into three pieces. When the deposits are large enough they are cast in what is called a shoe mould. These moulds are generally 15 in. long, 3½ in. wide, and 1¾ in. deep in the inside. The bar is called a "shoe bar," the mould a "slipper mould." It has two separations in it, which divide the bar into three pieces. A silver deposit bar from this mould would weigh about 200 oz. The weight of such a mould filled will generally be from 175 oz. to 225 oz. When the melt is not so large it is either cast in one or two divisions of these moulds, or else in small special moulds.

The deposit melter takes two samples of each gold or silver deposit. These samples are taken from the "shoe bars" by chipping a fragment from the diagonally opposite corners, if there is but one bar; if the melt makes several the assay sample is taken from the first and last bars of the melt. Usually when small melts of silver or Dorè metal are made, two samples are taken and granulated from the bottom of the pot, but in the case of very large melts the samples are taken from the first and last dip by taking about 200 oz. from the melting pot, pouring off all but a few ounces, and granulating 1½ oz. in cold water. Each of these assays is marked with the number of the melt, and with a mark signifying whether it came from the top or bottom of the metal.

After the gold and silver deposits are sent to the parting house to be parted they are no longer kept separate. The materials containing lead are put on one side and are several times a year cupelled on an English iron water-back cupel hearth. It is usually arranged that this work of cupellation shall be done during the winter, as the heat of the process in the confined space where the furnace is situated would be too great in the summer months.

The crucibles that are used are graphite, Nos. 8, 12, 20, 35, 90, 100, and 150.

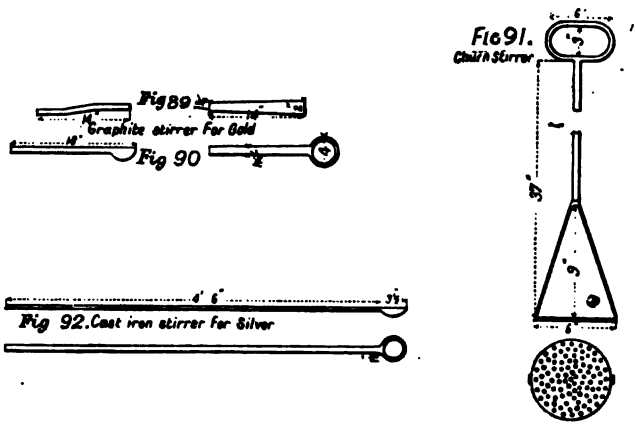
		oz.
No.	8 crucible holds...	200
,,	12 ,, ,, ...	250 to 300
,,	20 ,, ,, ...	400
,,	35 ,, ,, ...	1300
,,	90 ,, ,, ...	2000 to 2500
,,	100 ,, ,, ...	3500
,,	150 ,, ,, ...	6000

These figures do not represent the crucible full, but only what can be conveniently used in them. They last from fifteen to twenty melts, depending on the character of the bullion treated. Occasionally they are broken by accident, and to provide for this possibility a special arrangement is made of the fireplace of the furnace. If there is much lead in the bullion it corrodes the crucibles very rapidly. Previous to casting from any melt the contents of the crucibles are stirred. For this purpose there are two kinds of stirrers, one of graphite made in two forms, a straight one like the one at the Philadelphia Mint, Fig. 89, and a flat one with a circular termination, like the one used at the New York Assay Office, Fig. 90. For silver the stirrers are made of an iron disc perforated with holes, set in a stirrup to which a long handle is attached, Fig. 91, or a flat disc on a long handle, Fig. 92. The use of the graphite stirrer is greatly facilitated by having a wooden bench in front of the furnace so that the workmen seizing it with the tongs with both hands can work it from the shoulders, instead of, as formerly, from the wrist. This enables the men to work more freely and with less fatigue, and at the same time to make the metal more homogeneous. The motion given with the graphite stirrer is

rotary; that of the stirrup stirrer, is like that of a churn. Since the introduction of this stirrer it has been found that the melts are much more uniform than they formerly were. Besides the graphite crucibles, Hessian crucibles are sometimes used.

The loss in weight in melting varies according to the material treated. With ordinary bullion it ranges from 0.05 to 0.15 per cent. With amalgam the loss depends upon the degree of heat to which it has been subjected. It will generally not be more than 0.5 per cent., though it depends of course upon the quantity of mercury contained in it.

The melting furnaces that are used at the United States Mint and Assay Offices are all of the same general character. They



have been modified from year to year, having been originally introduced by Mr. Peel in the year 1835 as the result of a comparison of all the furnaces which were used in the mints in Europe. Nothing remains of this furnace except the outside shell, which is of cast iron. The furnace is so well adapted for the melting of the precious metals with the least loss that a description of it is given in full. Figs. 93 to 99 give a plan, end and front elevation, and sections on the lines C D and A B showing the construction of the furnace in detail, and Figs. 100 to 112 the details of the different parts.

The principle on which the furnace is built is to have the whole interior lined with cast iron in such a way that the fire-

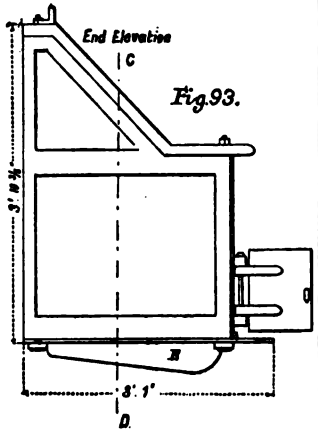


Fig. 93.

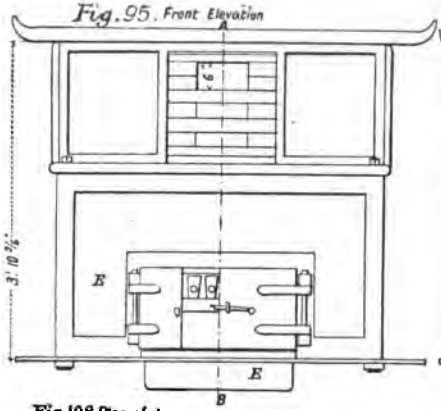


Fig. 95. Front Elevation

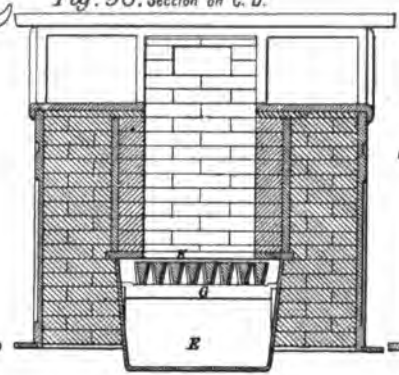


Fig. 96. Section on C. D.

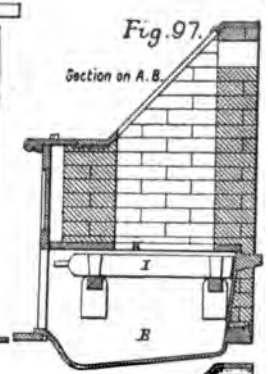


Fig. 97.

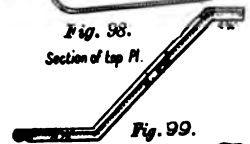
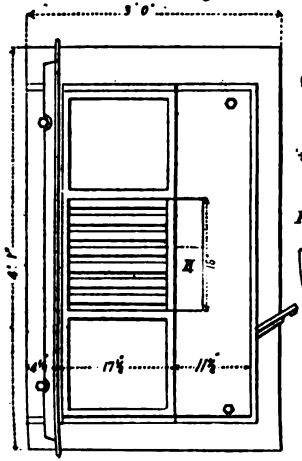


Fig. 98.



Plan Fig. 94.

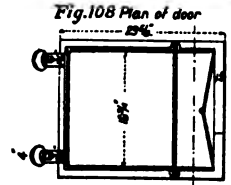


Fig. 108 Plan of door

Fig. 112. Flat Poker

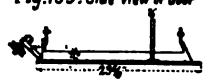


Fig. 109. Side view of door

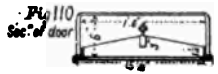


Fig. 110
Sec. of door

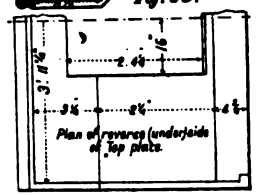


Fig. 99.

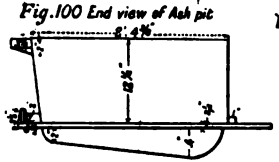


Fig. 100 End view of Ash pit

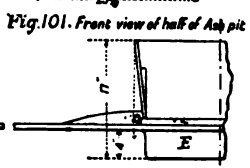


Fig. 101. Front view of half of Ash pit

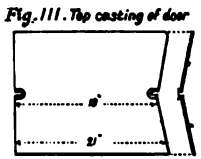


Fig. 111. Top casting of door

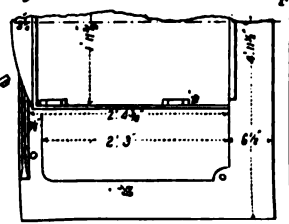


Fig. 102 Plan of half of Ash pit

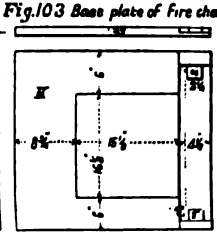


Fig. 103 Base plate of Fire chamber

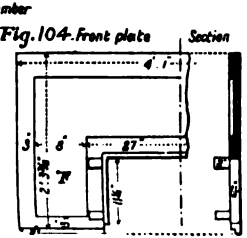


Fig. 104. Front plate Section

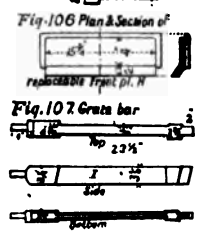


Fig. 107 Grate bar

Fig. 105 Grate bar bearings

Fig. 106 Plan & Section of

bricks can be taken out with the greatest ease, both for repairs to the furnace and for the chipping of the brick and of the grate bars to remove the precious metals attached to them. The space between the inside and outside lining of cast iron is filled with ordinary brick; the furnace is built of cast iron, having an outside and inside shell, as shown in the plan, sections, and elevations, with a sliding door for the top of the furnace, shown in detail at Figs. 108 to 111; this door is lined with firebrick, has a handle upon it, and runs upon rollers in such a way that it can always be moved with the least friction and effort, no matter how hot the furnace is.

The outside plates are shown in plan, elevations, and sections. The section on the line C D, Fig. 96, shows the outside of the furnace built with red brick, with a cast-iron central frame, against which the refractory brick is placed with its cast-iron supports on the bottom, Fig. 97, and the position of the grate bars which are also shown in the section A B, and in the Figs. 105 to 107, which show the grate bars and their supports. The cast-iron lining includes an ash-pit, Figs. 93 to 97, and 100 to 102, which is detached from the rest of furnace and is made entirely of cast iron. It is 4 in. below the level of the floor in front and 2 in. behind, and reaches 12¼ in. above the level of the floor, so that the ashes can be taken out with a shovel. It is 2 ft. 4⅜ in. deep. The object of this construction of the ash-pit is to facilitate the gathering of the ashes daily, so as to allow of their being treated occasionally when it is convenient, and not allowing them to accumulate, and also to prevent the gold and silver from getting into the bricks of which the ash-pit is built, which they did as it was formerly constructed. The back of the ash-pit is made heavier at the top and the bottom than the sides, Figs. 96 and 97, as the top coming near the grate bars is liable to be burned, and the bottom is likely to be cracked from the heat. It has two projections on it at the back to fasten it securely to the brickwork. Towards the top of the ash-pit are the grate bars I, Figs. 97 and 105 to 107, which are 3¼ in. and 1 in. wide on the bottom, 2½ in. wide at the top at the ends, and 1½ in. wide in the middle, leaving an inclined space between each bar. The bar on which they rest in front is shown

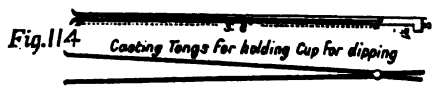
at G, Figs. 97 and 105. Above this is set the bedplate of the furnace K, Figs. 96, 97, and 103, and on this the back and sides of cast iron, Fig. 96, to receive the lining of refractory brick. The sides of the bedplate are cast in one piece, but the front part, as it is liable to burn out, is made separate, so that it can be replaced. This front will last for three or four weeks, and is easily replaced in part of a day. The surface of the bricks of the furnace are picked every day for grains of gold and silver. The men are so anxious to get out all the gold that these bricks are roughly treated, and do not last more than a few days.

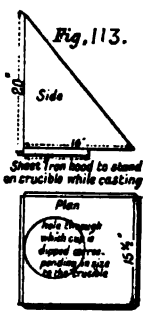
The cast-iron bedplate is replaced once or twice a week, but the ash-pit may last for a very long time. The grate bars, Fig. 107, are arranged so that there is a space between them and the back part of the ash-pit, Fig. 97, so that the air will pass up to cool them and increase the length of their life. The front of the bars is reduced in section, Fig. 107, so that they can be easily withdrawn. The working door, Figs. 108, 109, 111, and 112, is of peculiar construction and extremely useful on account of the ease with which it moves on the rollers, and also because the firebrick lining is covered over with sheet iron so that a current of air can pass between them, and when the furnace is at its hottest the door can be moved readily without burning the hands. A small section of the cast iron in front of the furnace towards the middle of the fire, Fig. 97, is liable to be burned out, and a repair piece, Fig. 106, is arranged for this. The ash-pit doors shown on the front and end elevation and plan are so arranged as to control the draught in the furnace perfectly.

The arrangement of the front plate is shown in Fig. 104. The section of the top plate and a plan of its underside is shown at Figs. 98 and 99. The whole frame is easily put up with bolts. The work of lining it takes but a short time, and any repairs can be very expeditiously made. The draught can be regulated by means of a damper in the chimney, so that whenever it is necessary either to dip or to add fine material to the melting pot, it can be done with the least possible danger of loss of the precious metals.

Fig. 113 shows the sheet-iron hood which is placed over the top of the crucible whenever a charge is to be introduced into

the crucible or dipped out from it. This hood fits over the top of the pot and prevents anything which might be mechanically carried off or volatilised, from passing up the chimney, while it does not interfere with the draught. It is very light, easily handled by the men, and protects them absolutely from the fire. A flat poker, Fig. 112, is used for raking the fire. The tools which are used for stirring the silver are the two cast-iron stirrers, one with a long handle and a round disc, Fig. 92, and one with a perforated disc, Fig. 91. The graphite stirrers for gold are shown, the one used in the Philadelphia Mint at Fig 89, and the one used at the New York Assay at Fig. 90. The tongs used for the casting cup are shown at Fig. 114, and the casting cup itself at Figs. 115 and 116.





The arrangement of this furnace is such that it works with but little fuel, and requires but few repairs. There is also the least amount of iron and silicious matter exposed to the gold, and consequently there is much less of these materials to be treated subsequently to get the gold out of them, while, as shown by the drawings, the furnace is so made that it can be easily repaired and the iron collected when it is necessary to treat it.

A full description of the very original methods by which the precious metals are recovered from the iron is given in detail further on. The fuel used is generally Lehigh anthracite coal if the melt is to be a heavy one, and charcoal if it is small, which is found to be so advantageous that it is transported even to San Francisco for the purpose.

The heaviest weight that can be conveniently handled for casting is about 120 oz., so that it would require fifty dips to

cast 6000 oz. of gold and about forty dips for 3600 oz. of silver. This removes all the metal except a small amount of liquid material to start the next melting. This frequent dipping requires to be very carefully done, and demands a great deal of skill on the part of the men, as the openings of the moulds are all narrow, and the number of pourings and stirrings to secure uniformity of composition and prevent liquation require that the surface of the metal to be dipped should be about 30 in. from the floor. This is the most convenient height for the workman to use his strength economically and with the least fatigue, as the casting lasts from ten to fifteen minutes. In case there should be any spilling of the metal every arrangement is made to catch it.

Gold and silver at their temperature of fusion are more volatile than is usually supposed. They become all the more so if there is a strong draught passing over their surface, or if they are associated with any amount of volatile metals. For this reason a cover is always placed over them, and the draught used in the United States mints and assay offices is rather a uniform draught from a tall chimney than one produced by a blower. Every time that any stirring of the metal is done the dampers are completely closed. In order still further to prevent losses while the metal is being charged into or dipped out of the large crucibles, they are covered with the hood, Fig. 113. The surface is kept constantly covered with borax or powdered charcoal during the melting, and the damper is kept closed during the entire time of casting.

In all the mints of the United States the melting furnace which has been described was exclusively used until 1887. The cast-iron sides and ash-pan make the loss in gold and silver appreciably smaller than it was before the iron linings were invented, and there is a great deal less of silicious material to grind to regain the gold and silver either volatilised or attached to the brick. Since the year 1887 gas furnaces have been introduced for the deposit melting, and in some of the offices even for bar melting. It is likely to supersede the use of coal.

The fluxes which are used in melting are, borax with most melts; potassium nitrate, if there is anything required to be

oxidised, such as iron or carbon, or lead in the metal; sodium bicarbonate if retorted amalgam is being treated; bone ash as a cover and to thicken liquid slags in order to skim them; corrosive sublimate in some of the mints when the bullion contains large amounts of lead or antimony; phosphorus is used in quantities not exceeding half an ounce in a melting containing 2000 oz. when the bullion contains copper. Powdered charcoal, prepared for the purpose, as at the Philadelphia Mint, is put over the surface of the metal to protect it from the air. In all the mints the hood, Fig. 113, is put over the pot at the time of charging and at the time of casting, both to protect the men and to prevent loss by volatilisation. Usually the melt does not come within 6 in. of the top of the crucible, in order to prevent loss in this way. In order to remove lead, a covering of bone ash ¼ in. thick having been put over the surface, crystals of soda nitre are put in through holes made in the covering, and after the effervescence has been allowed to go on for some time, but not entirely stopped, the ordinary blacklead stirrer, Fig. 89, which is held in the tongs, is moved in circles round the top to spread the nitre, and is then plunged down to the bottom of the metal, moving it up and down. The pierced stirrer, Fig. 91, is not used for gold as it would render it impure, but is always preferred for silver. The graphite dipper is used because it allows of lifting the cupful of metal up and then pouring it back, thus giving a greater motion to the vessel. These operations are performed very quickly. The surface of the melted metal is skimmed with the triangular crucible, the flat side being down, so as to take off only the flux from the upper part of the metal. This operation is done so rapidly that the exposure to the air and the carbon of the vessel is not sufficient to reduce the foreign metals. When the silver looks right, or when a test shows by bending that it is right, it is cast.

After the deposits have been melted in the deposit room, and the assays have been taken, if they contain both gold and silver they are sent to the parting house to be quartated or parted. Before quartation, however, they are inquartated so as to have at least two to two and a half parts of silver to one of gold. It has been found by experience that when the gold is in less pro-

portion than this it separates in a fine powder which frequently floats on the surface of the acid during the parting, but by having a comparatively large quantity of gold it will become coarse and is easily separated. In most of the mints the metal is granulated, but it is found that the shoe bars that are 900 fine in gold and silver dissolve easily as bars without granulation and without breaking them. In order to ascertain the fineness of the silver when it is to be cast into ingots a cupful containing about 200 oz. is taken from the top of the melt. All but 5 oz. or 6 oz. is put back into the crucible, and the remainder is granulated in a small copper cup and marked No. 1. These cups stand generally in two rows, No. 1 being behind. At the last of the melt another similar assay is taken marked No. 2.

As soon as the report from the assay shows that the gold is fine it is sent to the minting room to be refined. Mint fine gold averages 997 gold and three silver. The coin standard gold is made by adding copper, the law prescribing that the amount of copper in standard coins should be not less than 9 per cent.

The moulds which are used for the different coins are described in the description of the Philadelphia Mint. The average weight of the gold ingot melts is about 3000 oz.

It has been found even when the metal is thoroughly stirred before casting the ingots, that there is a tendency for the silver in them to be richer in the centre than at the sides. It has been found that the central portion of a rolled strip of standard silver may sometimes be from .0005 to .001 finer than the edges of the strip. The planchettes are therefore very frequently finer than the clippings. Allowance is made for this in making the coin. The gold and copper alloy does not act in the same way.

The gold and silver of commerce, when it is melted, is very often brittle, nearly as brittle as glass, so that the simple striking of a light blow on a piece of soft wood is sufficient to break a bar of it in two. Before it can be used for coining or is fit to use for chemical purposes it requires to be purified, and this process of purification is called toughening. As the Mint takes everything that comes, it gets material, especially from old jewellery, which may contain tin, lead, and zinc, besides the usual impurities. In addition to these it may contain iron—a

very small quantity of which may render the gold extremely hard—arsenic, antimony, and in fact anything which great skill, thorough dishonesty, and rapacity can invent for making an article look like gold, but containing as little of it as possible. This gold if it contain silver must be inquartated in order to be subjected to the process of quartation or parting by acids. If the silver or gold does not contain any of these metals it is simply melted.

The losses of gold in melting are directly proportional to the quantity of volatile metals which they contain, to the length of time exposed, the quantity melted at one time, and the power of the draught which has access to the surface of the melted metal.

A melt of 5000 oz. of gold containing one-tenth of one per cent. of impurity would upon being melted give 4900 oz. of pure gold, and the residues would contain 95 oz. of gold, and a certain part of the 5 oz. of impurity which rendered the gold brittle. This would be 100 oz., which must be treated, 5 per cent. only of which was impurity, from which the precious metals are to be separated.

The charges collected from depositors for parting, refining, and toughening bullion at the coinage mints and the assay offices of the United States at the close of the year 1885 to 1886, being the excess of the charges deposited to the credit of the fund over and above the amount drawn from it for expenditure during the period of ten years commencing July 1, 1876, was $184,436.57.

The charges collected and the expenditures at each institution during the fiscal year 1885-1886 are given in the following Table:

CHARGES COLLECTED FOR PARTING AND REFINING BULLION AND EXPENDITURES DURING FISCAL YEAR 1886.*

Institutions.	Charges Collected.	Gross Expenditures.	Net Expenditures.
	$	$	$
Mint at Philadelphia	13,067.43	6,060.76	6,060.76
,, San Francisco	56,355.91	58,662.76	58,662.76
,, Carson	47.86	939.80	939.80
,, New Orleans	763.09	1,411,36	1,411.36
Assay office at New York	92,621.24	100,497.29	83,318.48*
Total	162.855.53	167,571.97	150,393.16

* The value of blue vitriol and spent acid sold during the year, previously credited on bills for acid, was $17,178.91.

THE PARTING PROCESS USED IN THE UNITED STATES MINT AT PHILADELPHIA.

The Nitric Acid Process.

The United States Mint at Philadelphia is a fireproof building, containing besides the works of the mint, properly speaking, large vaults for the storage of bullion and coin, and a museum of great historical interest. A parting process, invented in the year 1866, by Mr. A. Mason, the present superintendent of the New York Assay Office, but then its melter and refiner, is used for separating the precious metals. The same process is also used in the mint at San Francisco. It is carried out in both places very efficiently and cheaply. The process for both mints will be described together. This process is known as the double process as both nitric and sulphuric acid are used in it. It consists in inquartating the bullion, so that the gold shall constitute 100 parts in 285 of the granulated metal; this proportion leaves sufficient margin for any want of homogeneity that might occur in the granulated bullion, while the porous character of the resulting metal causes the acid to dissolve the silver more completely, so that not more than 6 per cent. of silver is left in it. The residuum from the nitric acid is boiled twice with sulphuric acid, and makes gold 998 fine. This process is applicable to all kinds of alloys, and has been successfully worked since its invention. It is much cheaper than the old nitric acid process, requiring very much less fuel and 20 per cent. less nitric acid. The sulphuric acid part of the process is also very cheap, and requires but little apparatus and few men.

A brief summary of the work of this mint during the years 1884-85 is given below. There were coined during the year $18,509,280.25 of gold and silver. This value was contained in 42,864,328 different coins. The amount of gold coined during this time was 8,880,322 oz., with a wastage of .002 of a standard ounce. The amount of silver treated in the same time was 16,330,992.87 oz., with a wastage of 2148.83 oz., 1980.12 oz. of which were stolen, so that the actual wastage of the year was 168.71 oz. The amount of gold and silver parted was $3,381,962. The weight of the gold in gross ounces was 288,747, in standard ounces it was 153,390. Its value was $2,853,766. The

weight of the silver in standard ounces was 453,919, and its value was $528,196. The charges collected from depositors for the parting were $12,188.02. The expenditures for doing it were $8250.97, leaving a profit of $3937.05.

In the year 1885-86 there were coined 24,997,460 pieces of a value of $27,660,039.40. The smaller number of pieces being owing to the falling off in the demand for minor coins. The gold and silver together coined consisted of 23,290,809 pieces of the value of $27,642,661.75. Besides this there were struck 10,038 nickel pieces of 3 and 5 cents, and 1,696,613 bronze cents. The excess of this year's work was mainly in the bronze pieces.

The melter and refiner treated 1,396,820,218 standard ounces of gold with a wastage of 421,603 standard ounces, and 39,693,151.81 ounces of silver and reported at the end of the year a surplus of 4384.42 standard ounces. The same officer operated on 2,604,286.24 oz. of metals used in the minor coinage, with a wastage of 4,456.12 standard ounces.

The coiner operated on 1,118,730.030 standard ounces during the year, with a wastage of 24,541 standard ounces. The same officer operated on 39,005,873.90 standard ounces of silver bullion during the year, with a wastage of 3,413.63 oz.

The work of the melting and coining departments of the Mint at Philadelphia during the year 1884-5 was on a scale never before attempted. The total amount melted and refined was:

	Tons.
Coinage of gold	42¾
,, silver	1215
Refining gold and silver	18
Total	1275¾

All of this gold and silver was handled sixteen times while passing through the melter and refiner's hands, and fully as often in the coining department, so that upwards of forty thousand tons of gold and silver were handled by the operatives of the Mint during the year.

The parting process used consists of eight operations. These are:

1. Receiving and melting the deposits.
2. Inquartation and granulation of the material containing gold.

3. Solution of the granulations in nitric acid.
4. Boiling the gold residues twice in sulphuric acid.
5. Precipitation of the argentic nitrate with salt.
6. Reduction of the argentic chloride with zinc.
7. Sweetening and pressing the silver into cakes and drying the gold and silver.
8. Casting the silver and gold into ingots and bricks.

I. Receiving and Melting the Deposits.

The United States Mint receives upon deposit, bullion in bricks and bars from all parts of the United States, and besides, parts all articles of gold or silver, which it receives on deposit, to be assayed and their value ascertained. Large quantities of old jewellery, and of new jewellery out of fashion, of all kinds, watch-cases, and any other materials containing gold and silver, are received. Anything containing a sufficient amount of the precious metals is taken on deposit, except that, so far as is possible, material containing tin is rejected. This is the case with much of the cheap jewellery, which is filled with solder, and with watch-cases, which, within the past few years, have frequently been alloyed with tin, which is a difficult metal to separate with the appliances which are used at the Mint. It might be melted with lead and cupelled, but there are no cupellation furnaces at the Mint, and the refining by fluxes is very difficult, and gives a great deal of residue. Each deposit is melted by itself, and, if large enough, is cast in the slipper moulds, if not, into any kind of form. This deposit when melted is numbered, assayed, and sent to the melting-room, where it is all mixed, so as to make the alloy of proper composition for parting, if it contains gold.

The deposit melting-room contains three furnaces of the same size as all the others in the Mint. They use, however, charcoal as fuel. The gold is melted in Gunlack-Hessian crucibles, holding about 200 oz. each. The silver is melted in blacklead crucibles, Nos. 20 to 35, holding from 1000 oz. to 1200 oz. each. The whole work of this department is done by three men and one superintendent. They use about 100 bushels of charcoal per week.

In melting the bullion a cover is almost always used. The crucible is so large that there is very little danger of the draught ever reaching the melted material. This cover is generally of charcoal or of bone ash. When the material contains metals which are to be oxidised, the cover is made of bone ash $\frac{1}{16}$ in. thick over the melted metal, and then nitre is added through holes made in it to oxidise the impurities.

According to the quality of the bullion, the cover is renewed from time to time as is required. If it is very impure it may be renewed three times. Usually it is not done more than once or twice. When the metal is sufficiently purified the cover is carefully removed and the metal fluxed with borax. The cover and the slag are ground, sifted, and washed to recover the small particles of bullion. All the material from the bullion deposit room is toughened. If it is not sufficiently tough on the first melting, it is remelted with pulverised charcoal.

Many of the silver bars which come from New Mexico and other localities, contain considerable amounts of arsenic and antimony. This metal is refined by putting into the melted bullion an iron bar 4 ft. long, 3 in. wide, and from $\frac{1}{2}$ in. to 1 in. in thickness, with which the metal is briskly stirred. A little nitre is added to the charge, and the metal is dipped with an iron ladle. This will generally remove all the antimony after two or three minutes of stirring. Before this method was devised it was found extremely difficult to separate the antimony from the bullion. When tin is present, pearlash is used, and thoroughly mixed with metal. If the slag is too thick a little borax is added. When iron and sulphur are present in either gold or silver, a flux of one part sand and two of nitre is used. Bone ash is used to thin the slag. When lead is present in either silver or gold, bone ash is used. Nitre is added, and holes made through the cover from time to time as it seems best.

The fluxes which are generally used in melting are sand, borax, sal ammoniac, soda, salt, and nitre. Sand is generally used when the gold or silver bullion contains iron. Borax is used when the metals are quite pure: sal ammoniac when the metal contains tin, or other substances difficult to separate. Nitre is used as a general oxidising reagent, especially when the metal contains

lead. When the slags are thick, soda is used to thin them, so as to make them deposit the granules of metal, and, if this is not sufficient, salt is added to it. Whenever any of these fluxes are used the metal is not allowed to come within 6 in. or 7 in. of the top of the crucible. When the metal is entirely fused it is left for some little time in quiet fusion, and then stirred with a graphite stirrer held in the tongs, so as to give the whole force of the upper part of the body, in stirring, to make the metal homogeneous, and not depend upon the wrists. Since this method has been introduced, very few melts have been found to be unequal in value at the top and the bottom of the ingot. As soon as it is stirred it is poured. When cast, the deposit is sent into the deposit room to have its number marked upon it, and, subsequently, to be assayed to determine its value and fineness.

In the deposit melting room graphite crucibles, Nos. 12, 20, 35, 50, and 70, are used. These usually last a week, or about one hundred melts. Of the Hessian crucibles, which come five in a nest, only the two largest ones are used. They last for five or six melts. Anthracite coal is used for large melts and charcoal for small ones.

When the deposits are received they are melted just as they come with fluxes and poured. If tough they are sent away at once. If not, they are melted under borax and charcoal until they are. Sometimes they require to be melted twice, and at others three times, until they are perfectly tough.

II. Inquartation and Granulation of the Material containing Gold.

The melting furnaces are on the first story directly under the parting room. Here all the gold and silver is melted either for granulation or for ingots, which are the strips to be rolled out to make coin. The room is exactly the same size as the parting house, and is divided into two parts by a chimney, which has four furnaces on each side of it, and three at one end, making eleven in all, each of which takes a No. 70 graphite crucible. They melt 3500 oz. of standard silver and 6700 oz. of standard gold. The silver bars weigh from 20 oz. to 180 oz., and the gold from 100 oz. to 200 oz. The crucibles last thirty-six

meltings of silver and twenty-four of gold. Ingot melting is done every day of the week. Granulation is done only once a week, the metal being made of three parts of silver and one of gold. The granulation is done in water, which is cooled with ice. It is done in a copper vessel with a pan like that used in the New York Assay Office, with three handles caught by three hooks. The back hook is very short so as to throw the pan forward to one side which has holes at the bottom in the rim on the front side, so that the water can drain off. When the granulation is finished the water is drawn off with a copper syphon, which connects with a pipe in the floor leading to the sewer, and fresh water added. To effect the granulation a wooden platform one foot high is placed in front of the furnace. The metal is dipped out of the crucible held high above the head, and poured in a thin stream with a slight wavy and rotary motion. The doré silver is not granulated, but is sent to the parting-house in ingots or bricks.

III. Solution of the Granulations in Nitric Acid.

The granulations are carried from the melting-room to the parting-room, which is on the floor above, in wooden boxes lined with copper, which hold 2500 oz. They are 18 in. by 13 in. and 6 in. deep, with two iron handles upon the outside for lifting them. The transportation is done on low platform wagons with iron wheels, which carry twelve or more boxes, and are raised to the level of the parting-house by a lift. The granulations are poured into a flaring copper pan 40 in. by 33 in. on the top and 24 in. by 30 in. across the bottom, and 8 in. deep. The rim is reinforced by a ¾-in. half-round copper bar. It has two handles to carry it by, and, if necessary, to lift it when full. It is raised by a Weston pulley-block. The pan is set upon a table 25 in. high, which is directly against the wall next to a pair of platform scales, on the same level as the table. The granules are taken out of it with a copper scoop and put into a copper receiver on the platform of the scales and weighed, so as to get the exact quantity for each pot. The room in which the parting is done is lighted on both sides and one end by large windows, but is not very light, owing to the proximity of other buildings. It is 80 ft.

long, 30 ft. wide, and 12 ft. 6 in. high, with a stone floor. The corroding house is in the centre of one end. The pots are contained in a wooden house 23 ft. long, 10 ft. high, and 5 ft. 6 in. wide, the bottom of which is raised 12 in. from the floor. The vertical sides of this house are 7 ft. high. The roof is inclined; the peak of it is cut flat for a foot, and is two feet below the ceiling. The inside of this has a double lining of boards, which can be taken out when it is too much eaten. The whole house is painted on the inside with asphalt, and on the outside with ordinary paint. On the bottom it is divided into two parts, each of which has two vats. The larger of these houses receives six parting-pots on a side, and the other five. Each section of the house has two tanks lined with 12-lb. lead, on the side of which, three inches from the wall, is a special plank used as a splash-board, to prevent the throwing up of the water by the steam when it is boiled. Each of these tanks is 2 ft. wide, and the length of half the house. Its depth is 22 in. They are made of 2-in. plank. On the bottom of each of them a pipe is placed for the circulation of the steam to heat up the water.

In each compartment of the house there are three doors on a side, which give access to two pots each, except the last door at the end of the smaller one, which gives access to only one. These doors are 40 in. wide and 45 in. high, and are made in wooden frames, the corners of which are reinforced by irons bent at right angles. They are counterpoised by weights hanging upon chains from each side, so as to allow the door to be opened to a height of 5 ft. 9 in. above the ground, which is necessary to give the requisite space for handling the wooden stirrer used in the pots. Each one of these doors has two iron handles for the purpose of lifting it easily. The first house has all the doors alike; it is 12 ft. 6 in. long. The other is 11 ft. 6 in. long, and has two doors alike, and one of 28 in., which gives access to only one pot. The whole house is braced with joists, which are 4 in. by 3 in. between each door. They rise a few inches higher than the bottom of the doors when they are opened.

All four of the tanks have water on the bottom to the depth of 10 in. This depth is regulated by an overflow pipe, so that it is always the same. These overflow pipes are in the centre of the

side of the house, and the steam for boiling comes in at the two ends.

The pots used are made of glazed earthenware. They come from Germany. Most of them are 21 in. in diameter and 22 in. deep. They vary a little, however, the limits being 21 in. to 24 in. in diameter and 21 in. to 22 in. in depth. Each of these has two conical handles on the outside, which are 6 in. wide and 5 in. high. Each has a spout 2 in. wide at the lip and 1½ in. deep. The weight of the corroding jars is 90 lb. These are made in Germany, and do not crack by any heat that is produced by steam, and are only broken by accident. They were formerly made in this country, but caused so much loss by cracking, and were so easily broken, that only German ones are now used. They last for a long time, most of those on hand at the present time having been purchased during the war. They are set in the vats on a lattice-work of wood 1½ in. thick, to protect the bottom of the tank from injury, with holes 4 in. square. This rack is set between the steam pipes which run around the sides. These pipes are of copper, 1½ in. in diameter, and make the round of the tank next to the four sides of it. They are pierced on the insides with ⅛-in. holes every 2 in. The pipes are thus behind the splash-board, the steam being discharged towards the interior of the tank. A wooden paddle, 33 in. long, with the flat 3 in. wide, 1 in. thick, and 9 in. long, is used in each of these pots to stir the mixture. The flue for the whole house is at one end. It is 29 in. on the bottom and 1 ft. high vertically. It then follows the slope of the roof so that at the highest point it is 2 ft. A flue from the chimney for cleaning the blanks for making the copper coins enters the side next the flue. From the brickwork the flue is continued in a terra cotta pipe, 14 in. in diameter, and set in a brick flue bound with iron. It descends vertically 30 ft., and enters a horizontal covered culvert, 4 ft. by 3 in., which is about 40 ft. long, and connects with a chimney 8 ft. 6 in. on the inside and 120 ft. high; 125 lb. of granulated bullion, a part of which, in some cases, consists of small silver coins which are condemned, are put into each one of these pots.

The granulated bullion accumulated during one month will fill from twenty-two to twenty-eight pots. The charge of granula-

tions in each pot is 135 lb. Five pitchers of nitric acid, which each hold 25 lb. of the acid at 40 B., are put into each pot, covering the bottom nearly 9 in. The whole height of the charge in the pot, including granulations, will be 12 in., which about half fills it. Steam is introduced into the pipes on the bottom of the tank and the water brought up to a boil and kept so for twelve hours. In order to prevent them from getting clotted the granulations must be stirred every twenty minutes, each stirring taking about a minute for each pot. No cover is placed over the pots. They are left uncovered for greater ease of stirring. The sliding door for this purpose is raised, but the draught is so strong that none of the fumes come out into the room. The only acid fumes come from the transfer of the various acids. The workmen leave the building at six o'clock, but the steam is kept up until 8 p.m., at which time it is shut off. During these two hours the house is looked after by the night watchman. The water to thin down the argentific nitrate is not added until the following morning, when the bath is heated up for a few minutes to dissolve any crystals that may be formed. The water is added until it reaches to within 3 in. of the top of the corroding pot. The solution is then drawn off with a gold syphon $1\frac{1}{2}$ in. in diameter; this has a taper wooden plug $2\frac{1}{2}$ in. by $1\frac{3}{4}$ in. The syphon costs $3600. The silver nitrate is collected in wooden tubs 18 in. in diameter and 15 in. high, made with staves $1\frac{1}{4}$ in. thick. They are held together by two hoops of 2-in. iron, and have handles on the inside for the purpose of carrying them. They last for two years. When the acid is drawn down to near the top of the gold three pitchers of fresh acid are added, and the granules are boiled for another twelve hours as before. Each charge thus remains in the pot forty-eight hours, during half of which it is constantly boiled. This last acid is not saturated It is baled into an adjoining empty pot in the house, a leaden strip being placed between the two, so as to prevent any spilling into the water bath. This acid is used on a fresh charge.

The pitchers for the acid are of the same shape as the ordinary bedroom pitchers, but much larger. They are 18 in. high, 6 in. in diameter in the opening above, and 9 in. at the belly. Besides the usual handle on one side, they have

also one on the belly to lift them with. They are made of ordinary enamelled earthenware. They weigh 10 lb. each when empty. The nitric acid is brought to the lift in carboys. These are carried on wagons which hold two each, and turned at once into earthen pots 18 in. in diameter and 28 in. high. From here it is drawn into the pitchers. There are four of these receivers. Five pitcherfuls are counted to a carboy. When the acid is baled down so that there is danger of taking out the gold, water is added to dilute the rest of the acid, and it is syphoned into a wooden tub on the floor outside the parting house. When the acid is dilute enough, the pot is lifted out of the house and the gold washed out of it into a filter. This filter is a round wooden tank, made of staves $1\frac{3}{4}$ in. thick; these are reinforced on the outside with others $2\frac{1}{4}$ in. wide by 1 in. thick, leaving a space of $1\frac{1}{2}$ in. between each outside stave. The whole is bound with three hoops of 2-in. iron $\frac{1}{8}$ in. thick, and is painted on the outside. The filter is 28 in. in diameter on the inside at the top and 24 in. at the bottom. It has a false bottom, 4 in. above the permanent one, filled with $\frac{1}{4}$-in. holes, leaving 17 in. for the inner height of the filter. The false bottom is first covered with a layer of cotton cloth, which is brought up over the top and tacked to it. One thickness of Swedish filter paper, which takes three sheets, is put over the bottom. Thick stiff brown paper is then put round the sides. Then a thickness of muslin, which, like the paper, is carefully pleated on the sides, is tacked over the top of the tub. The filter paper is thus between two thicknesses of muslin. This filter has a wooden cover made in three parts. The whole filter is set inside of an iron frame resting upon a wagon; the frame is so arranged that the four arms by which it is caught shut over the wooden cover and make it quite tight, and form two eyes at the top by which it is lifted. Between the false and the real bottom there is a large wooden stop-cock, with a $1\frac{1}{4}$-in. hole projecting 12 in. from the side, and supported by two irons hooked to the inside of the filter. Into this filter the gold in the bottom of each vat is poured, and the pots thoroughly washed to take out the last traces of gold. It is then raised to the overhead crane and carried to the large silver tank, where it is suspended and the gold then carefully washed with hot

water, which enters from an iron pipe in the ceiling, and is allowed to run by itself until it is perfectly sweet. This takes from two to three hours.

The charge from eleven pots will be about 1 ft. high on the bottom of the filter. When the gold is quite sweet the tank is lowered to the ground and the gold is removed from the cloth by earthenware dippers to be treated by sulphuric acid. The upper muslin of the filter lasts for two operations, the lower one lasts for three. When they are no longer strong enough to bear the weight of the gold they are dried and burned, and the gold recovered added to the other gold.

IV. Boiling the Gold Residues Twice in Sulphuric Acid.

The gold from the filter is carried to four furnaces, which are hexagonal in shape, 25 in. high by 28 in. across. These are made of boiler iron with cast-iron doors and tops, and are lined with brick. Anthracite coal is used in them exclusively. The cast-iron cover is 1½ in. thick. The top has on the outside a rim ½ in. deep. In its centre is a hole 21 in. in diameter with an inclined flange 2 in. wide, having the same angle as the bottom of the sides of the pot in which the gold is to be treated. This rim rises ½ in. above the top, so that there is a basin ½ in. deep all around the top of the furnace to catch anything that may be spilled over the sides of the cast-iron pots. This hole has a cover 1½ in. thick. In the top of this furnace a cast-iron pot, with three legs 3 in. long, is placed. It is nearly hemispherical, being 22 in. in diameter at the top and 12 in. deep in the centre. It has two trunnions. When set in the hole of the furnace 7½ in. of the pot are above it. The trunnions prevent its passing in any further. It is intended that the bottom of the pot should fit the flange pretty nearly tight. With these trunnions, when the operation is finished, the pot is lifted by two men, by placing a piece of gas pipe over the trunnions, so as to give a handle long enough to get an easy purchase. The lid of the furnace is then slid under for the pot to rest on. It was originally intended to lift these pots by the trunnions with a Weston pulley-block and carry them away on an overhead railway, but it has been found easier to lift and carry them by these pipe handles. The pots

are closely covered by a conical lead hood, with a pipe 6 in. in diameter made of 12-lb. lead sliding in another 8 in. in diameter. The hood has a hole in the side 7 in. by 6 in., which is covered with a piece of sheet lead bent over at the top, so that it can be easily removed when its contents are to be stirred. These hoods are counterpoised with weights so as to be easily lifted; 8 ft. above the furnace is a reservoir 2 ft. square, the length of the four furnaces, and lined with 12-lb. lead. It is inclined to one corner. The 8-in. pipe enters the bottom and rises 2 in. above it. In the centre of the box is a flue leading to the terra cotta pipe in the ventilating flue of the parting house. Whatever acid condenses in the box flows to the corner and is carried by a lead pipe to a vessel 7 in. in diameter and 12 in. high on the floor. The rest of the acid escapes into the flue.

The gold is dipped out of the corroding pots with a porcelain vessel 6 in. in diameter and 4 in. deep with a porcelain handle 5 in. long. Under this a porcelain vessel 10 in. in diameter and 2 in. deep is held to catch any gold that might possibly fall. Ten of the porcelain scoopfuls are put into each of the iron pots. One pitcherful of sulphuric acid at 66 B. is then added in each pot. This acid weighs 32 lb. It is boiled for one and a half hours. By this time it has nearly evaporated. A second pitcherful is put on and boiled for the same time. The pot must be stirred every ten or fifteen minutes, which is done through the opening in front, with an iron rod the end of which is hammered out into a four-pronged rake, the prongs being 3 in. and the whole rake 3 ft. long. This is done to prevent the gold from agglomerating on the bottom. By this time the gold is 996 to 998 fine. If wanted finer than this it is boiled a third time, and can easily be made 999½ fine. The excess of acid is poured off to be used again. The acid from the gold is put into a lead-lined vat 20 in. in diameter and 24 in. deep, lined with 8-lb. lead, bound with three iron hoops and diluted with water, and the silver contained precipitated in the porcelain pot. The residue of lead and silver sulphate is allowed to collect. The gold after being carefully washed with water is kept in a porcelain solution pot under water. All the wash water from the gold is treated in the same pot. When small samples are treated the gold

residues are filtered on a frame 2 ft. square, the inside of which is 20 in. square; a single thickness of muslin is tacked to this, and over the muslin two thicknesses of filter paper are placed. The gold is washed down the sides of the filter paper with a two-quart pitcher. The liquor is saved and added to the silver liquors. When sufficient gold has been collected it is carried to the drying furnace in the cellar, which is similar to the drying furnace for silver, but a little smaller. It is not pressed into cakes, but when dry is sent directly to the melting-room.

The sulphuric acid is stored in a vat made of 2½-in. wood, 23 in. by 19 in. by 20 in. deep. This is covered inside and out with 12-lb. lead, and holds 2½ carboys, which do not fill it quite to the top. The acid is dipped out of this with a two-quart pitcher. The large pitchers are set upon a lead plate 2 in. larger in diameter than the bottom of the pitcher to make sure of catching any acid that may drop from the outside. The tank and the pitchers are set upon a platform 6 ft. long and an inch wider than the carboy, with a rim 2 in. deep, lined inside and out with 12-lb. lead. The tank is set at one end, and the pitchers at the other. Each pitcher is set upon a lead basin so arranged as to allow the pitchers to drain perfectly from any acid that may be spilled on the outside.

V. Precipitation of the Argentic Nitrate with Salt.

All the nitrate solutions from the eleven pots go to a large tank to have the silver precipitated. This tank is 10 ft. by 8 ft., and 4½ ft. deep. It is made of 3-in. staves, which are reinforced on the outside by others 3 in. wide by 1 in. thick, with the space of 3 in. between each outside stave. This is held by six hoops 2¼ in. in width of ⅛-in. iron. The tank is set 3 ft. above the floor. It is 4 ft. deep on the inside, and 4 ft. 6 in. deep on the outside. The salt solution for precipitating the silver is made in a tank on the floor above, the water being allowed to take up as much salt as it will dissolve. The salt for this solution is brought up in bags which contain one bushel each. The solution is conveyed to the tub in iron pipes and is run into the tank until it is 1 ft. deep on the bottom. The whole is stirred by a wooden dasher 2 ft. wide by 1 ft. high, with a handle 3 ft. longer than the

longest diameter of the tub. It takes two hours to bring the solution from the corroding jars and thoroughly precipitate the silver from it. After six pots have been added the liquor is thoroughly stirred. A platform is placed on the outside across one of the longer diameters of the tank, which is 6 ft. high and 7 ft. long, 18 in. wide in the centre and 34 in. at the sides, being cut out to fit the shape of the tub. The top of the platform is reached by six steps on wheels, 6 ft. wide, which are rolled out of the way when the silver chloride is carried to the reducing tank. On the opposite side is a platform 28 in. by 32 in., hung to the sides by iron bars, which can be moved to any part of the tank when required. It is 58 in. above the ground. A ladder is also used to reach any part of the tank when it is necessary. When the test shows that the silver is all out of the liquor it is again thoroughly stirred with a dasher and the heavy parts of the precipitate pushed to the side of the tank where the spout is on the long diameter of the tank. The whole of the liquor will be about 14 in. deep on the bottom of the tank. The spout by which it is emptied is 18 in. long, 4 in. in diameter, with a discharge hole $2\frac{3}{4}$ in. in diameter. It is supported by two iron rods attached to the side of the tank. This spout projects over a tank resting on a wagon, the top of which is 2 ft. above the bottom of the floor. This tank is 6 ft. 6 in. long, by 2 ft. 7 in. wide, and 15 in. deep. It is lined with 12-lb. lead. It is set upon a frame on the top of the wagon, which is 3 in. thick and 9 in. high. This top inclines to the centre and also to one end, leaving a space of 3 in. at the lower end between the top and bottom of the tank. The bottom of the tank is filled with holes $\frac{1}{4}$ in. in diameter, and over these and the sides of the tank twilled cloth is put and kept to its place on the sides with a frame made of $\frac{1}{2}$-in. stuff painted. Both the frame of the filter and the wagon top have their corners reinforced with iron. The silver chloride tank is independent. It rests upon a cleat on the large frame, which is securely fastened about 2 in. from the bottom of the frame. The top of the truck inclines towards the centre and one end. Under the centre of this end troughs made out of a single stick 7 in. wide and $4\frac{1}{2}$ in. thick, hollowed out in the middle, are placed. The one next the tank projects a few

inches under the bottom of the tank. The thickest of the silver chloride on the bottom of the precipitation tank is pushed by the dasher to the stop-cock and made to flow on to the filter till it forms a bed 2 in. or 3 in. in thickness. This is called a bottom. After this has been done the rest of the contents of the tank is made to flow over this bed, which is thick enough to prevent any fine silver chloride from passing through the cloth. While the silver chloride bottom is being formed the liquor does not run through the filter clear, and low wooden pails, 13 in. in diameter and 8 in. high, with copper hoops and handles, are set under the trough to catch it. As long as it is turbid this liquor is thrown back into the vat. When a glassful held against the light is perfectly clear, the pails are removed and a trough like the others, but 6 ft. long, is placed under the bottom trough and carries the excess of liquor away. The silver chloride is washed for about four hours with cold water until sweet. It is tested both by colour and by paper to see when it is perfectly sweet. The water runs through two or three traps and then goes to the sewer.

VI. Reduction of the Argentic Chloride with Zinc.

The wagon, with the silver chloride in the tank upon it, when it has been completely drained is moved to a tank between the precipitation tank and the gold-refining furnaces. This tank is 6 ft. long by 3 ft. 9 in. wide and 18 in. deep. It is lined with ¼-in. lead. This lead lining lasts for two or three years and then gives out in the seams. The corners of the bottom of this tank are rounded for greater ease in stirring. Four boxes 20 in. by 12 in. and 6 in. deep made of ½-in. wood with two iron handles on the outside, filled with granulated zinc, are thrown into the tank to reduce the silver chloride. These will generally be sufficient, but occasionally four and a half boxes will be required. The calculation is generally made that it will take two boxes for eleven solution pots, but it sometimes takes a little more. The zinc box weighs, when empty, 10 lb., and when full of zinc 42 lb. so that each box contains a charge of 32 lb. of zinc.

The silver chloride is shovelled out of the vat on wheels with a copper scoop shovel. The chloride is put into the tank first, and the zinc stirred through it afterwards. The action of reduc-

tion should commence at once; but if it does not, a pint of sulphuric acid is added to make it begin. It is stirred with a wooden stirrer, 4 ft. 4 in. long, with the shovel part 10 in. by 17 in. and 2½ in. thick, the end of which is cut down to a sharp edge. One man must stir it very actively for an hour. After this it need only be stirred occasionally. It takes four hours to finish the whole. When the reduced silver is very dry a little water is added to dissolve out the zinc sulphate.

This vat was formerly covered with a lead-lined hood counterpoised, and having a chimney, 14 in. square, going to the roof. This was raised from time to time as needed in the working. It was very heavy, and the men eventually became afraid of it, so it was taken down. No inconvenience is felt from the gases evolved. The charge in the tank is 12 in. deep on the bottom. When there is no further action three pitchers of sulphuric acid, weighing 32 lb. each, are added to dissolve out any excess of zinc. The vat is allowed to lie all night with the acid in. The reduced silver is pushed to one side and any liquor in excess is drawn off with a copper syphon.

The zinc is granulated in a copper tank like the bullion granulating tank. It is 32 in. in diameter at the top, and 27 in. deep and slightly flaring. It has two handles on the side. A pan, 26 in. in diameter and 6 in. deep, fits into this. It has three handles, two at the ends of one diameter, and one at the diameter at right angles to it, and is in all respects like the silver granulating tank. The zinc is melted in a crucible in a furnace exactly like the melting furnaces, and is poured into this tank with a rotary motion. A ¾-in. pipe discharges cold water into the tub; a 1¾-in. iron pipe, 15 ft. long, carries the escaping water to the sewer. When the pan is full it is raised by a pulley. The handles cause it to tip to one side. It is suspended over the top of the tank to drain, the front of the pan being perforated so that the water flows back into the vat. The zinc from the pan is thrown into a wooden, hopper-shaped reservoir on wheels. This is 35 in. by 30 in. on the top, and 40 in. high. On the bottom there is an opening 1 ft. high the whole width of the box. In front the bottom extends into a shelf 16 in. wide. The granulated zinc is removed from here with a shovel. One of these cars full lasts for four reductions.

VII. Sweetening and Pressing the Reduced Silver into Cakes.

The reduced silver is taken out with a copper scoop 10 in. in diameter, and 5 in. deep, with a shovel handle, and put into a colander. This is a cylindrical wooden vessel on wheels, lined with 8-lb. lead, which is turned over at the top. It is 3 ft. in diameter, 20 in. deep to the false bottom, which is 6 in. above the other. It has four $1\frac{1}{2}$-in. iron hoops. There is a 1-in. overflow pipe at the top and running down the sides. The silver here is washed with hot water until it is entirely sweet.

Before putting in silver the lead bottom, which is filled with $\frac{1}{4}$-in. holes, is covered with one thickness of twilled cloth, and is filled to within 4 in. of the top. The water runs out as fast as it runs in. This water flows into a box with two partitions, and from there to the sewer. These boxes are simply used as a precaution. Not more than 2 oz. or 3 oz. of silver per year is caught.

The nitre used in the melting was formerly made in the mint from the nitric acid used to brighten nickel coins. This was saturated with soda under a wooden hood, and concentrated at a very low heat in an iron vessel lined with lead, and crystallised for use in the melting department.

The articles contained in the parting house are:

22 German corroding jars.
6 American ,, ,,
7 Pitchers for nitric acid.
2 Earthen reservoirs for nitric acid.
1 Lead tank for sulphuric acid.
2 Two quart pitchers for sulphuric acid.
7 Wooden stirrers.
1 Porcelain jar for the wooden stirrers.
1 Wooden tub for copper scoop and shovel.
2 Silver nitrate tubs.
2 Salt trucks.
7 Silver chloride dripping tubs.
2 Sulphuric tubs for cleaning gold.
2 Large tub filters for gold.
1 ,, ,, silver.
1 Silver chloride tank.
2 ,, reduction tanks.
2 Zinc vats on wheels.
4 Boxes to measure charge of zinc.

6 Carboy trucks for square boxes.
1 Iron truck for round carboys.
1 Copper scoop.
1 ,, shovel.
1 Porcelain scoop and basin.
4 Iron stirring rakes for gold.
2 ,, scrapers to clean up the floor.

Every time a colander is filled and washed, it is carried to the hydraulic press in the cellar. The silver is put into a circular mould 12 in. in diameter and 12 in. deep. The plunger of the hydraulic machine fits this mould rather loosely. The silver is patted down with a round wooden stamper 4 in. in diameter and 4 in. high, with a handle 5 in. long. The plunger is then made to descend until the cake is only 4 in. in thickness. About a gallon of water is forced out of each cake. This water formerly ran through six traps, but it is now caught on a twilled muslin filter; the water runs from it directly to the sewer. The silver is dried in the same kind of a furnace as the gold, but it still contains water, so that it shrinks considerably in drying. The shrinkage amounts to nearly 2 in. in the diameter. In November, 1885, ten colanders were sent to the press. With six men three runs of eleven pots are the most ever done in a week. The business of the mint is much less than it formerly was, for in 1848-9, during the gold excitement in California. fifty-six pots were treated in a week.

The gold-drying furnace is 3 ft. by 6 ft., and 4 in. deep. It is made of $\frac{3}{8}$-in. boiler iron, and is covered by a rectangular hood. From the top of this hood, in the middle, a pipe, 12 in. square, conducts the steam to the chimney which carries off the smoke. The front of the hood inclines a little, so that the top is slightly smaller than the bottom. It is 22 in. high. It is hinged so that it can be raised to give access to the whole of one side of the furnace. In the centre of this door there is a small one 11 in. square, which is used for observation during the process. An iron shovel and a copper scoop are used both to introduce and to remove the gold. It contains very little dust, as the gold from the sulphuric acid is generally coarse, and there is no fear of loss in melting, as the hood over the melting-pot stops the draught. The gold is not pressed into the bricks, but charged into the pot as such.

The silver-drying furnace is 7 ft. by 38 in. and 4 in. deep, the pan being made in the same way as the gold pan. It holds twenty cakes. The flue of the fireplace in both of these furnaces passes through the centre of each, and directly under the pan. The ash-pit is 2 ft. deep. The wood used is pine, in sticks 3 ft. long. There are no grates to the fireplace, but the end of the wood rests upon iron in front, and the brickwork at the back, which is protected from abrasion by an iron plate. The gold and the silver are brought in deep copper trays, set upon wagons. The trays for the gold are 39 in. by 20 in., and 16 in. deep. For the silver they are 36 in. by 24 in., and 18 in. deep. The silver cakes are piled up in the bottom, two in a row. There will thus be three rows, of six each, piled one on top of the other, and in the centre at the top two, care being taken, as far as possible, that they should not touch the copper sides. The cakes, as they come from the furnace, are shrunk unequally; those in the centre, directly over the flue, are the most contracted. They are often brought up to a red heat, and sometimes contract as much as 2 in. in the diameter. Those upon the sides frequently do not contract at all, or are very much less contracted than those in the centre. The furnace is fired while some other work is being done. It requires very little attention. The fireplace is 14 in. square, the wood is lighted with shavings put into the ash-pit and allowed to burn out, when another charge of wood is put in. It usually takes three firings, with the furnace full of wood, to heat the silver, and one and a half for the gold.

VIII. Casting the Silver and Gold into Ingots and Bricks.

In the ingot melting-room the furnaces, Figs. 93 to 112, are arranged two upon one side against the central flue, with their coal-bins. One of the furnaces is used for granulation. There are, besides these, two others at the side of the room. In the next room there are three against the chimney and four at the end of the room on the main wall, making eleven in all. The crucibles used are of graphite. They are set in the furnace on half fire-bricks made for the purpose. When the ingots or bricks are too long for the pots, graphite rings, 3 in. deep, are put over the top of the crucible with a cement of borax and silica. This

melts and fastens the ring securely to the pot, making it perfectly light. When the melt is finished, just before the last dip is taken from the crucible, this rim is pried up with a poker, and is easily removed. The pot is then used as before. These rings can be used three days.

When a pot is to be charged all the material is brought in a copper-lined box 18 in. by 13 in. and 6 in. deep. The part of the melt in small pieces is taken up first in a copper scoop. The larger pieces are then lifted upon the scoop by the hand. The weight of all silver melts for coin is 3000 oz. The weight of the gold melts varies according to circumstances. An iron hood, Fig. 113, with two handles to lift it, is then placed over the pot, the round part of the bottom of which is the size of the outside of the top of the pot. The front fits over the incline of the front of the furnace, so as to effectually shield the men from the fire and prevent any danger of dropping a part of the charge into the fire. Usually there will be a little metal in the bottom of the pot from the previous melt, which cannot all be dipped out. The small pieces are added very carefully at first, and the larger ones as carefully on top of them to prevent any splashing. When the metal is in brick form, too large to go to the bottom of the pot, the cover is put over the brick on the door side, inclined so as to protect it as far as possible from the air. The fire is kept up as actively as possible. In order to prevent the air from coming in by the sides of the door, bars of iron are placed against it.

A melt of 49,000 oz. of bullion divided into twelve melts occupied a day and a half, there being less than 1000 oz. of silver and litharge and other oxides in the skimmings. These consisted of the bone ash made pasty by litharge, and the flux used, which was in more or less of a hard sponge, the silver being mostly in the shape of grains entangled in it.

The skimmings from the crucibles are charged into pots with some charcoal as a reducing agent, and pearlash to thin the slags. This was melted at a bright-red heat and allowed to settle quietly, so that the metals made a "king" in the bottom with the slag above it. The slags are ground and sifted to recover the metallic grains. The "kings" are put into a crucible and gradually heated to a full red heat, the metals which liquate

during the time being collected separately. At first the material will be nearly pure lead; finally the precious metals will separate. What is remarkable in this process is the very small loss in lead. There is also a very small loss of the precious metals. At the Philadelphia Mint, where it has not been practicable to erect a cupel furnace, this process has been found by experience to work as well as cupellation.

The late Professer Booth suggested that where small quantities of silver contain zinc, tin, and antimony, it will be sufficient only to add lead to the metal to purify it from these metals which are so easily oxidised, and he has made a number of experiments for that purpose.

A melt of 75,000 oz. of gold which was extremely brittle, and which for toughening was divided into fourteen melts of 5400 oz. each, took a day and a half. These were melted in a graphite crucible with soda ash and fused borax, using 1 oz. or 2 oz. to each melt, the crucible being filled to within 3 in. of the top. When the metal was perfectly quiet, the slag being somewhat viscous and slightly puffy, 1 oz. or 2 oz. of saltpetre were dropped into the centre of the metallic surface through the flux. Their mixing with the metal was aided by the circular motion of a small crucible. The moment the oxidation ceased to be visible to the eye the slag was skimmed off with a dipping crucible as quickly as possible, so as not to take up any metal; the remaining melt was quite tough. It is worthy of remark here that a single part of foreign material in this melt was sufficient to make the whole of the metal brittle, this being composed of good standard gold, which contains only 10 per cent. of copper. The loss in gold was hardly appreciable, and the copper, which is so easily oxidised, was not affected, the amount remaining exactly the same afterwards as before. The height of the metal in each pot was from 9 in. to 10 in., and the oxidation took place readily from the top to the bottom, the object of stirring being only to make the metal uniform throughout. This experiment was a particularly characteristic one, as the amount of impurity contained was extremely small. When it is larger the exact fineness of the metal can be easily ascertained by casting a thin strip and testing it under the hammer.

The men acquire such skill from long habit in handling large quantities of commercial gold that they are very frequently able to tell by a simple inspection of a fractured surface not only the quantity, but also the quality of the base metal which produces the brittleness, so as to be able to add the proper quantity of soda, borax, and nitre. It is, however, always to be noticed that with such melts as these the alkalies attack the crucible wherever they touch its sides, sensibly weakening it. The larger amount of the materials used in fluxing which has to be removed, also renders it likely that some of the metals already oxidised in the flux may be reduced and returned to the metal, which makes it more difficult to purify, so that the operation may have to be repeated once, twice, or even three times. In order to provide against the danger of possible loss under these circumstances, with large quantities, as the slag is very liquid when it is necessary to remove it, it is thickened with sand, lime, or bone ash, which is stirred into it so as to make it possible to skim without fear of the metals being reduced. As the action here is a powerful mechanical one, the men generally prefer sand. There is, however, a danger in using them on standard gold, there being likely to be a decrease of fineness of the gold by a removal of some of the copper, so that a brittle gold, which is 900 fine, may sometimes after toughening be 900.05, or even 901.

From the entire toughening of the 75,000 oz. there were less than a crucible full of skimmings. These are melted and liquated, the slag is ground and separated into metallic grains, and a poor cinder, too poor to use, which goes to the sweeps. The grains and the "king" are melted and fluxed as before. The "king" for this operation weighed only 8 oz. Professor Booth estimated that the entire impurity which caused this brittleness in the whole 75,000 oz. was less than an ounce.

The following Table* gives the number of melts and ingots made and the number condemned at the Mint of the United States at San Francisco from 1874 to 1886:

* Report of the Director of the U.S. Mint for 1886, p. 26.

Fiscal Year.	Gold Ingots.		Silver Ingots.	
	Number Made.	Number Condemned.	Number Made.	Number Condemned.
1874 . . .	813	5	2,648	10
1875 . . .	925	13	4,378	15
1876 . . .	942	6	9,454	11
1877 . . .	1,141	3	13,210	8
1878 . . .	1,393	19	13,610	14
1879 . . .	981	4	12,789	14
1880 . . .	931	3	8,104	14
1881 . . .	1,033	8	12,617	38
1882 . . .	958	8	10,719	20
1883 . . .	901	5	7,509	12
1884 . . .	767	4	5,539	1
1885 . . .	677	1	2,619	0
1886 . . .	935	0		
Total . .	12,397	79	103,196	157
Condemned per cent.	...	.6	...	.1

The time of melting is usually for silver one hour and ten minutes, for gold one hour and forty-five minutes, per melt. Anthracite coal is used exclusively in this department. When the metal is quite melted it is thoroughly stirred just before casting. For silver this is done with the round iron disc, Fig. 91, 6 in. in diameter and $\frac{1}{4}$ in. thick, pierced with eleven rows of $\frac{1}{4}$-in. holes. This is set in a stirrup of $\frac{1}{2}$-in. iron, the attachment being made to the side of the disc only. This stirrup is 9 in. long, and is attached to the rod. The whole length inside the handle is 37 in. The handle is bent to an oval, and is 3 in. by 5 in. It is moved up and down in the melted silver from the top to the bottom of the crucible. After some use this disc becomes bent, and the iron is eaten into by the metal in a very irregular manner, but always in the direction in which the iron has been rolled. These stirrers last, when in constant use, ten days. For gold the stirrer is made of blacklead, Fig. 89. It is 14 in. long, $1\frac{3}{4}$ in. wide, and 1 in. thick on the upper part. The lower part gradually widens until at the bottom it is 3 in. wide. At this point there are three holes, two below and one above, $\frac{1}{4}$ in. in diameter, which go through the stirrer. The upper part is straight, but the lower part is slightly curved like a table

fork. The graphite stirrers for gold last three to four days. They are usually broken by accident, and do not wear out. They are held in a pair of pincers. The men step up on the wooden platform in front of the furnace, and seize the tongs with both hands, stirring the metal in the pot with a movement from the shoulder, in this way the work of stirring the metal is done most effectually with the least labour. The cups for dipping the gold and silver, Figs. 115 and 116, are also held in tongs when in use. They are made of graphite, expressly for this purpose, and last two or three days. They do not wear out, but are usually broken by accident. The pot covers last three or four days.

Connected with each fusion furnace are the following articles:

1 Iron scoop for charcoal.
1 Iron coal-bin and shovel.
1 Iron scoop to hold the metal.
1 Iron scoop, 10 in. by 6 in. and 3 in. deep, with handle 6 in. long, for collecting the scum taken from the pots.
1 Iron shoe mould to cast the metal in case of accident.
1 Pan for granulated silver, 18 in. by 12 in. by 6 in.
1 Hood for charging the crucibles.
1 Flat poker, 3 ft. long by 1 in. wide and $\frac{1}{4}$ in. thick, turned up 1 in. at the end.
1 pair of tongs to remove the grate bars.
1 Spring tongs.
1 Differential pulley for two furnaces.
1 Duster.

Each furnace man has a pair of crash mittens, made very heavy, and stuffed with old carpet, so as to be able to handle everything hot.

Two assays are made from each pot: the first after the first ingot has been poured, when some of the metal is granulated in a cup. The granulating cups are made of copper, and hold about a pint of water. Twenty-four of them are set in a rack in two rows of two, and are filled with cold water. The hind one is marked with chalk No. 1, and the front one No. 2. The samples are marked to correspond. The number of the melt is also chalked upon the cup. The first melt is marked No. 1. The second is taken out just before the last ingot is poured, and is marked No. 2. If the assay shows that the metal is of a proper fineness, the ingots are sent to the coining department; if not,

they are remelted. When the assay sample has been taken by chipping, the chips are put into a copper pan, which is marked in the same way.

Assay samples for standard silver are alone done by granulation. The assay samples for gold and fine silver are taken by chipping from the down corner. No doré metal ingots are made at the mint.

In thirteen hours six tons of standard silver ingots are cast, which keeps six furnaces occupied. The others are used either on gold or on granulations. The work is somewhat intermittent, but is quite severe while it lasts.

Including all the men of the deposit-room and parting-house, there are thirty-eight men employed in this department. In the melting department alone thirty-two men are constantly engaged. These men are occupied as follows:—

8 Opening and closing moulds.
7 Melters.
4 Fillers.
3 Men in the parting-house.
3 Men in the deposit melting-house.
3 Grinding cinders, &c., in the cellar.
2 Tapping ingots.
2 Picking up and pickling ingots.
1 Foreman in the deposit melting-room.
1 Foreman in the coin melting-room.
1 Overseer.
1 Collier.
1 Gatekeeper.
1 Man at the hoist.

The loss in weight in the treatment is estimated at one-tenth ounce for every thousand ounces. The melting of standard silver is done under granulated charcoal, entirely free from dust, made for the purpose. On fine gold and silver the charcoal cover is only used to prevent the action of the air. Fine gold, which is base, is first mixed with fluxes, and charged in the pot when the hood is on it, and is melted very slowly. The fluxes used for separating the different materials from gold are as follows: When arsenic is present the gold is melted under a bone-ash cover, and a hole about 2 in. in diameter is made in the centre, nitre being poured in around the circumference after the metal is in quiet fusion. When this is saturated it is removed. It generally takes two or three hours to remove the arsenic; but it can be done in this way. When lead is present the same is done, but it takes less nitre, as the bone ash absorbs most of the lead. When antimony is present the metal is liquated and the

antimony separated in the same way as the arsenic; but every cup taken from the melt must be tested. It is best removed by cupellation with wood as a fuel, the flame striking the metal. When tin is present bone ash and nitre are used. All the dampers in the flues are closed while working with the stirrer. The quantity of nitre added has to be very large.

For the impurities in silver, when antimony is present, the metal in fusion is stirred with a large piece of iron for about five minutes. When arsenic is present bone ash and a little nitre are used. The cover is allowed to stand until the bone ash flakes, when it is removed and fresh bone ash added until there is no flaking. For lead, bone ash is used alone. When tin is present, in silver, if there is any gold, it concentrates in the gold, and is separated in the same way.

The whole floor of the melting-room is covered with an iron grating full of hexagonal holes. The grating is in hexagonal pieces, 12 in. across, which are easily movable. They cost 8 cents a pound. Every three months it is taken up and the floor swept, and the silver and gold which would otherwise be lost, recovered. Beside the coin that is made for circulation, silver bars 999½ fine, which vary from 20 oz. to 180 oz. in weight, and gold bars, from 5 oz. to 243 oz. in weight, with a fineness of 999, are made. These are stamped with their fineness, weight, and value, and pass current just as the larger bars and bricks made in New York.

The ingot moulds are made of cast iron, in three pieces, planed up so as to have two ingot moulds in the same piece. They are clamped together with a strong iron screw, so that they can be easily separated. All three of the parts are greased with lard, put on with a swab, before they are used. The lard is purchased by contract, free from water. The kegs containing it are stored in the basement of the building, and what is used is held in a copper box 12 in. by 6 in. and 4 in. deep, with two handles at each end. At one of the ends there is a space 4 in. long, and the depth of the box, to hold the grease. The bottom at the other end is raised to within 2 in. of the top. The grease, taken from the deep part of the box, is spread over this inclined bottom, and the grease taken by the swab from here is spread over the mould. This is done so

as not to get too much grease on the swab. Standing at right angles to each furnace is a table made of wood, whose top is 2 ft. by 18 in. It is 18 in. high, with two projecting handles at each end at the top for the convenience of carrying the table when it is necessary to move it. On the top of this is a cast-iron pan to receive the ingot moulds, which are set on the bottom in two rows. These pans are 4 ft. by 2 ft. on the top, and 3 ft. by 14 in. on the bottom, and 6 in. deep. Their object is to catch any silver or gold that may be dropped from the cup during the casting. The sides flare to an angle of about 45 deg. This table, like all the other heavy tables, is set upon full hexagons of iron, the same size and thickness as the grating, so that when the floor is to be cleaned, these heavy tables need not be moved. The hexagons are 1 ft. from side to side. Eighteen double-ingot moulds are placed upon the table at one time. As soon as the mould is filled it is removed by a man, wearing very heavy gloves, to a table in front. This table is 8 ft. long, 2 ft. wide, and 30 in. high. The top is made of wood 4 in. thick, covered with a plate of cast iron 2 in. thick, with four rims running the full length of the table, $\frac{3}{4}$ in. wide and 1 in. high, the object of which is to keep the hot moulds and the ingots off of the body of the cast iron, and thus prevent its becoming heated. One man brings the moulds and opens them on the same side of the table as the furnace. On the opposite side a man with a copper tongs seizes the two ingots from each mould and places them on the end of the table until three moulds are open. He then seizes the six ingots in the copper tongs and plunges them into a rack in a bath of weak acid. This is contained in a box 22 in. square and 16 in. deep, made of 2-in. wood, and lined with 18-lb. lead. On the bottom of this box upright copper bars, set in a rectangular frame, are placed. This box is covered with a hood. A quart of commercial sulphuric acid is put into it. The hot ingots cool off in this pickle, the six together. When cool they are removed by the same man and put into a box, 24 in. by 18 in. and 10 in. high, made of sheet copper, into which cold water runs in and out to wash off the sulphuric acid. The top of this box is 24 in. above the floor, and on the same level as the pickle box, and at a convenient distance below the

table. It has the same kind of a rack as the pickle box into which the ingots are set. The small amount of silver in the pickle acid is precipitated, washed, and taken to the parting-house. After the ingots have remained in the water for a few minutes they are removed and put into a wooden box in front of the copper tank, and from there are collected to be prepared for the rolls. There are two of these cooling and pickling vats in the same room. As soon as removed these ingots are taken to a bench where they are caught in iron vices protected with copper plates over the jaws. The jaws are opened by a treadle commanded by the foot. The edges of the ingots are filed off roughly. There are four of these benches, with two men at each bench, two benches being in each division of the large room. As soon as filed they are collected, and go to a shear press to have the upper end of the ingots cut off, leaving the lower one wedge-shaped as it came from the mould, to facilitate its entrance into the rolls. They are then weighed, and if the assay shows the metal of proper quality they are sent at once to the minting department.

The ingot moulds are made of various sizes for the different denominations of coin to be made. They are all double except the quarter-eagle, which has four divisions. All these moulds are 13 in. in length, and are wedge-shaped at one end. Their width only is variable. For the $20 gold piece they are $1\frac{1}{2}$ in. wide and $\frac{1}{2}$ in. thick; for the $10 gold piece they are $1\frac{1}{8}$ in. wide; for the $5, $\frac{15}{16}$ in.; for the $3, $\frac{7}{16}$ in.; for the $2.50, $\frac{1}{10}$ in., and four to a mould; for the $1, $1\frac{1}{4}$ in. For the silver, the $1 is $1\frac{7}{8}$ in. wide; the 50 cents, $1\frac{3}{8}$ in.; the 25 cents, $1\frac{1}{8}$ in.; for the 10 cents the dollar ingot is used, two rows being cut out of it.

The melting and pouring of the silver dollars takes $1\frac{2}{3}$ hours, of the 50-cent pieces $1\frac{3}{4}$ hours, of the 25-cent pieces 2 hours. In these melts there are 74 dollar ingots, eighty-five 50-cent, 126 twenty-five cent, for the silver. For the gold the $20-pieces have 60 ingots to a melt, and take $1\frac{1}{4}$ hours; the $10-pieces have 96 to a melt, and take $1\frac{1}{2}$ hours; the $5 and the $3-pieces 150 to a melt, and take $1\frac{2}{3}$ hours; the $2.50-pieces 250 to a melt, and take 2 hours; the $1-pieces 96 to a melt, and take $1\frac{1}{2}$ hours. The $20 gold ingots rough weigh 75 oz.; the $10, 50 oz.; for the

$5, 40 oz.; the $3, 40 oz.; the $2.50, 20 oz.; the $1, 60 oz. The $1 silver ingots rough, about 65 oz.; the $0.50, 45 oz.; the $0.25, 30 oz.

The different denominations of coin have each their own special benches. Each of these benches has eighteen moulds. For the silver coins there are six benches for the dollar ingots, three for the half-dollar, three for the quarter-dollar, and for the 10-cent the same as the dollar. All the 10-cent pieces are made at the dollar tables. For gold there are two for double-eagles, two for eagles, two for half-eagles, eighteen for quarter-eagles, ten for $3 pieces, and thirteen for $1 pieces.

Below is the list of articles used in the melting-room:

36 wooden boxes lined with copper for alloy.
6 trucks for carrying alloy, &c.
36 alloy boxes, 17 in. by 11 in. by 4½ in., for the silver melts.
72 ingot boxes, 18 in. by 12 in. by 6 in.
60 granulating boxes, 20 in. by 15 in. by 5 in.
2 boxes for silver toppings.
1 box for gold toppings.
300 No. 70 graphite crucibles for silver and gold.
250 dipping cups.
250 No. 70 rings.
250 No. 70 pot covers.

As the iron of the furnace and the tools absorb a certain amount of gold, once a year, just previous to the final settlement of accounts in July, they formerly collected all the iron at the mint that might contain gold, in order to heat and oxidise it, and then beat off the oxide with hammers. By thus alternately heating and hammering the iron, all the precious metals were obtained in the iron oxide except what went up the chimney This required over a dozen men, during the heat of June engaged near the furnace in the melting room, hammering the hot metal which had accumulated during the year for three or more days of ten hours each, or 360 hours' labour to recover a small quantity of gold. As this was both a difficult and tedious operation, and very annoying on account of the constant noise of the hammering, Dr. J. C. Booth, the former melter and refiner at the Philadelphia Mint, substituted for it the conversion of all the iron into a sulphide, then dissolving the sulphide formed with dilute sulphuric acid. The residue being

gold and silver, was melted and parted. This was also found to be very tedious and expensive, and was abandoned. It has recently been discovered that the separation can be much better done by fusion, and Dr. Booth afterwards separated the precious metals from all the old iron, grate-bars, &c., which accumulate in the mint, by melting them. The materials are no longer allowed to accumulate in large quantities, 40 lb. or 50 lb. being treated as it is found necessary to renew the iron about the furnace. The amount of gold and silver contained in the iron is first determined by preliminary assay. The quantity of gold having been ascertained, silver is added until the metal contains two and a half parts of silver to one of gold. The whole is then heated to a white heat in a large black-lead crucible. Almost any kind of old iron materials, whether burned or not, can be made use of.

Since some of the iron that is used in this process is oxidised, and resists melting, or if melted is very viscous, and the precious metals settle through it with considerable difficulty, 50 lb. to 200 lb. of iron not burned, and especially very grey cast iron, with a large proportion of carbon, is added as a flux. If the iron is much oxidised a certain amount of charcoal must be added to reduce it. To this charge a certain amount of salt, borax, or a little soda, which is the usual flux in such operations, is added. The amount is not weighed, but generally about a handful has been found by experience to be the requisite quantity. When the crucible is charged it should be gradually raised to a white heat under the precautions which are generally observed in fluxing, and the whole charge maintained at a high heat perfectly fluid, for a number of hours, to ensure the complete liquation of the precious metals. To effect such a melting it is generally the practice to make it the last operation in the day's work, when the whole furnace is quite hot, so that during the night the charge in the crucible will have the opportunity of cooling very slowly, so as to produce a perfect liquation. In the morning the whole material will generally be found to be a solid mass. The crucible will contain, first, at the bottom the precious metals, gold and silver; second, above these the iron, which will be grey from its having cooled so slowly; and, third

the slag, which will be from ¾ in. to 1 in. thick, and more or less vitreous or stony, according as there is more or less silica present. On breaking the crucible the precious metals at the bottom are found separated perfectly from the iron. The cast iron has been found by a number of assays to be perfectly free from either gold or silver. I saw at the mint a lump of about 20 lb. of iron from which the precious metal had been separated, leaving a surface between the two that was almost smooth. It has been found that when white iron is used in the charge the separation is not quite so complete, as its fusion is pasty and a little gold adheres to it, while that of grey iron is liquid, so that when the grey iron is used no trace of gold can be found in the iron, the separation being absolutely complete. This separation being effected by liquation the longer the time that the metals can remain in a state of quiet fusion the better. The twelve or fifteen hours that it usually remains in the crucible makes the separation perfectly clean, neither metal entering into the other.

The experience of the Philadelphia Mint shows that the separation is more perfect when the quantity of silver is appreciably greater than that of gold, so that it is frequently found desirable to add silver to the gold, though in the first experiments the separation was made without any such addition. There is an advantage in adding silver, which is that the button of alloy obtained in the fusion can be at once separated by quartation, by treatment with nitric acid, and subsequently with sulphuric acid —the double process used at the Philadelphia Mint—without any further melting or preparation. The loss in gold and silver is so very slight that in the assays made up to the present time, no trace of gold has been found in the iron. A mass of 10 oz. of gold has been frequently separated from old iron in this way; there has occasionally been in the separation an apparent loss of ½ oz. But this has been found to be only apparent, as the detached gold grains in these instances made up the loss of weight of the button. The quantity of iron above the precious metals prevents loss by volatilization, while if the material remains long enough in a liquid condition the liquation of the metal has always been quite perfect. Instead of as formerly spending three weeks during the

heat of the month of June in melting and hammering all the iron which has accumulated during the year, whenever it is convenient, from 5 lb. to 50 lb. of iron are melted at a time, giving no extra trouble or labour and no appreciable extra expense.

There is always a considerable amount of loss in the melting of gold and silver, part of which is regained in the soot, a good deal from the floor, some from the iron and brickwork, &c., of the furnaces. The loss which occurs is given in the Tables below. It is much larger than is usually supposed, as the metal is melted three times before it goes into coin, and during each melt there is a small loss by volatilisation. The wastage or loss allowed by law is 1½ oz. of silver and 1 oz. of gold in 1000 oz., and it is always kept within the legal limits. It is quite uncommon in the practice of the mint that this loss, which goes by the technical name of "toleration," is attained. The amounts actually lost are considerably below these amounts. When it is considered that these metals must undergo at least two meltings, besides boilings in various acids, it is remarkable that the loss is so small.

The chimneys are cleaned once every year, during the last week in June, previous to making up the final settlement. The front of the flue to the ceiling is first taken down and carefully scraped. Most of the flue dust containing gold is to be found here. After the surfaces of the bricks are very carefully scraped, the doors and windows all being closed, a steel brush is run up and down the rest of the flue from the roof, the top of the flue being covered while this operation is being performed. All the material so collected is put through the Kent's amalgamator.

The following Table gives the wastage and losses on sales of these sweeps for 1885*. These losses are paid from surplus bullion, as far as practicable, and the balance reimbursed from the appropriations and silver profit fund:

Losses.	$
Melter and refiner's gold wastage	146.70
Coiner's gold wastage	50.81
Melter and refiner's silver wastage	1561.81
Coiner's silver wastage	3089.26
Loss on sale of sweeps	4593.04
Total	$9441.62

* Report of the Director of the U.S. Mint for 1885, p. 85.

Expenditures.	$
From contingent appropriation	483.69
,, parting and refining appropriation	457,24
,, surplus bullion	1491.24
,, silver profit fund	7009.45
Total	$9441.62

Losses.	Philadelphia Mint.	San Francisco Mint.	Carson Mint.	New Orleans Mint.	New York Assay Office.	Total.
	dols.	dols.	dols.	dols.	dols.	dols.
Melter and refiner's gold wastage . .	7,843.78	..	..	17.79	..	7,861.57
Coiner's gold wastage	456.58	..	..	..	..	456,58
Melter and refiner's silver wastage .	..	..	..	5,118 51	318.10	5,436.61
Coiner's silver wastage	3,413.63	..	..	2,037.80	..	5,451.43
Loss on sale of sweeps	5,760.54	2,564.32	1,640.87	2,406.67	4,764.85	17,137.25
Total . . .	17,474.53	2,564.32	1,640.87	9,580.77	5,082.95	36,343.44
Paid as follows:—						
From contingent appropriation . .	4,086.54	257.09	355.48	..	..	4,799.11
,, parting and refining appropriation	316.84	344.62	355.49	..	..	1,016.95
,, surplus bullion	6,188.89	1,839.99	..	448.68	5,082.95	13,560.51
,, silver profit fund	6,882.26	22.62	929.90	9,132.09		16,966.87
Total	17,474.53	2,564.32	1,640 87	9,580.77	5,082.95	36,343.44

The Table below gives the organisation of the mint at Philadelphia

GENERAL OFFICERS.

1 Superintendent.
1 Assayer.
1 Melter and refiner.
1 Coiner.
1 Engraver.
1 Cashier.
1 Chief clerk.
1 Assistant assayer.
1 Asst. melter & refiner.
1 Bookkeeper.
1 Weigh clerk.
1 Abstract clerk.
1 Cashier's clerk.
1 Warrant clerk.
1 Register of deposits.
1 Assistant weigh clerk.
1 Assayer's computing clerk.
1 Medal clerk.
1 Assistant bookkeeper
1 Register of warrants.

ASSISTANTS, WORKMEN, AND ADJUSTERS.

General Department.
1 Doorkeeper.
1 Messenger.
1 Painter.
1 Gasfitter.
1 Gatekeeper.
1 Helper.
6 Assistants.
5 Conductors.
6 Cabinet employés.
16 Night watchmen.
6 Carpenters.
2 Engineers.
1 Fireman.
1 Oiler.
8 Machinists.
2 Adjusters of scales.
3 Blacksmiths.
28 Labourers.
12 Female helpers.

Assayer's Department.
1 Assistant assayer.
1 Weigher.
1 Assistant.
1 Foreman.
1 Helper.
2 Workmen.

Melter and Refiner's Department.
1 Office clerk.
1 Foreman.
12 Melters.
4 Deposit melters.
1 Refiner.
2 Sweep cellar operatives.
1 Furnace builder.
1 Hoister.
1 Gatekeeper.
1 Labourer.
19 Helpers.

Coiner's Department.
1 Chief weigher.
1 Assistant weigher.
1 Hoister.
1 General foreman.
7 Counters (3 ladies).
29 Coining room employés (21 ladies).
10 Annealers.
9 Cleaners.
12 Rollers.
16 Cutters.
3 Medal room employés.
83 Adjusters.
51 Temporary adjusters.

Engineer's Department.
2 Engravers.
5 Diemakers.
1 Assistant.

The Table on page 730 gives wastage and loss on sale of sweeps for all the Mints of the United States in 1886. *

The Tables† on pages 733 and 734 give the bullion manufactured into standard silver dollars, and incidental wastage and loss on sale of sweeps at the coinage mints, from January 1, 1885, to December 31, 1885.

The following are the charges imposed at the coinage mints and assay office at New York on deposits and purchases of bullion.

Deposit melting charge‡ on all bullion, or coin not required to be parted or refined:

For each melt of 1000 oz. or less .	100 cents.
Over 1000 oz.	$\frac{1}{10}$ of 1 cent per oz.

PARTING AND REFINING CHARGES.

1.— *Parting Gold and Silver, or Refining Gold.*

(Rate per Ounce, Gross.)

On Bullion containing	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
	cents.	cents.	cents.	cents.	cents.
Under 200 M. gold . .	1½	1½	1	2	2
From 200 M to 399½ M, gold	2	2	2	3	3
,, 400 M to 699½ M, ,,	3	3	3	4	4
,, 700 M and over ,,	4	4	4	6	6
Over 100 M, base metal additional	1	1	1	0½	0½

In addition to the above charges, on deposits requiring parting or refining, for each deposit of 1000 oz. or less, a charge of 100 cents; over 1000 oz., one-tenth of 1 cent per ounce gross is made. For gold coin or standard gold bars the rate per ounce charged is imposed only on the number of ounces required to be refined to raise the whole to standard.

Silver allowed the depositor is calculated on the basis of refining the gold to 990 M.

* Report of the Director of the U.S. Mint for 1886, p. 100.

† Prod. Precious Metals in the U.S. for 1885, p. 224.

‡ Report of the Director of the U.S. Mint for 1885, p. 133.

II.—*Refining Silver.*

(Rate per Ounce, Gross.)

On Bullion containing	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
	cents.	cents.	cents.	cents.	cents.
Less than 897 M. silver .	1½	1½	1½	2	2
From 897 M to 979½ M, silver	1	1	1	1½	1½
,, 980 M to 997½ M, ,,	0½	0½	0½	1	1

In addition to the above, on silver deposits requiring refining a charge on each deposit of 1000 oz. or less, 100 cents; over 1000 oz., one-tenth of 1 cent per ounce gross is made. For standard or sterling bars the rate per ounce is charged only on the number of ounces required to be refined to raise the whole to the fineness of such bars.

TOUGHENING CHARGE.

Gold bullion	½ to 2 cents per ounce, gross.
Silver ,, ...	¼ to 1 cent per ounce, gross.

ALLOY CHARGE.

On the number of ounces of copper required to reduce the bullion to standard	2 cents per ounce, troy.

BAR CHARGE.

On bullion deposited for fine bars not required to be parted or refined, and for standard, sterling, or unparted bars:

Bars of	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
	cents.	cents.	cents.	cents.	cents.
Fine gold, per $100 value .	5	5	4	10	10
,, silver, per ounce, fine	0.¼	0.¼	0.$\frac{1}{10}$	0.¼	0.¼
Standard gold, per $100 value	10	10	10	10	10
,, silver, per ounce, standard	0.½	0.½	0.½	0.½	0.½
Sterling silver, per ounce, gross	0.⅓	0.⅓	0.½	0.½	0.½
Unparted bullion, per ounce, gross	0.½	0.½	0.⅓	0.½	0.½

The charges imposed at the Assay Offices, other than New York, on deposits of bullion, at the Mint at Denver conducted as an Assay Office, and the Assay Offices at Charlotte, N.C.: Boise, Idaho; Helena, Mont.; and St. Louis, Mo. are given on p. 735.

MINT AT PHILADELPHIA.

Month.	Manufactured.		Wastage and Gold in Sweeps.		Total Consumption.		Number of Standard Silver Dollars Coined.	Seigniorage.
	Standard Ounces.	Cost.	Standard Ounces.	Cost.	Standard Ounces.	Cost.		
		dols.		dols.		dols.		dols.
January	846,656.26	831,871.95	2,148.83	2,148.83	848,805.09	834,020.78	985,200	153,328 05
February	773,437.50	757,022.75	..	..	773,437.50	757,022.75	900,000	142,977.25
March	1,117,259 38	1,088,226.60	3,755.64	3,755.64	1,121,115.02	1,091,982.24	1,300,200	211,973.40
April	1,203,125 00	1,169,984.77	..	..	1,203,125.00	1,169,984.77	1,400,000	230,015.23
May	1,203,125.00	1,172,833.84	499.86	499.86	1,203,624.86	1,173,338.70	1,400,000	227,161.16
June	1,203,253.90	1,169,869.85	4,930.50	4.930.50	1,208,184.40	1,174,800 35	1,400,150	230,280.15
July	1,546,875.00	1,499,174.71	1,688.81	1,688.81	1,548,563.81	1,500,863.52	1,800,000	300,825.29
August	1,546,875.00	1,495,185.07	..	..	1,546,875.00	1,495,185.07	1,800,000	304,814.93
September	1,460,980.47	1 397,717.30	2,293.42	2,293.42	1,463.273.89	1,400,010.72	1,700,050	302,332.70
October	1,460,937.50	1,390,920 77	..	..	1,460,937.50	1,390,920.77	1,700,000	309,079.23
November	1,460,937.50	1,382,213.06	..	..	1,460,937.50	1,382,213.06	1,700,000	317,786.94
December	1,461,221.09	1,370,279.07	2,767.46	2,767.46	1,463,988.55	1,373,047.13	1,700,330	330,050.33
Total	15,284,783.60	14,725,305.34	18,084.52	18,084.52	15,302,868.12	14,743,389.86	17,785,930	3,060,621.66

MINT AT SAN FRANCISCO.

January	429,687.50	420,491.07	..	..	429,687.50	420,491.07	500,000	79,508.93
February	429,687.50	420,092.84	..	..	429,687.50	420,092.84	500,000	79,907.16
March	171,875.00	167,896.19	..	..	171,875.00	167,896.19	200,000	32,103.81
April	85,937.50	83,791.34	..	..	85,937.50	83,791.34	100,000	16,208 66
May	85,937.50	83,680.32	..	..	85,937.50	83,680.32	100,000	16,319.68
June	42,968.75	41,807.32	4,265.08	4,149.80	47,233.83	45,957.12	50,000	8,192.68
July								
August	40,390.62	39,261.23	..	..	40,390.62	39,261.23	40,000	7,738.77
Sepetmber								
October								
November								
December	..	..	744.51	718.91	744.51	718 91		
Total	1,286,484.37	1,257,020.31	5,009.59	4,868.71	1,291,493.96	1,261,889.02	1,497,000	239,979.69

MINT AT CARSON CITY.

Month.	Manufactured.		Wastage and Gold in Sweeps.		Total Consumption.		Number of Standard Silver Dollars Coined.	Seignorage
	Standard Ounces.	Cost.	Standard Ounces.	Cost.	Standard Ounces.	Cost.		
		dols.		dols.		dols.	dols.	do's.
January	85,937.50	84,441.53	..	..	85,937.50	84,441.53	100,000	15,558.47
February	85,937.50	84,196.71	..	..	85,937.50	84,196.71	100,000	15,803.29
March								
April	..	..	290.41	290.41	290.41	290.41		
May			..					
June	..	..	62.53	62.53	62 53	62.53		
July								
August	24,062.50	23,381.71	..	..	24,062.50	23,381.71	28,000	4,618.29
September								
October								
November								
December	..	..	1,495.78	1,453.46	1,495.78	1,453.46		
Total	195,937.50	192,019.95	1,848.72	1,806.40	197,786,22	93,926.35	228,000	35,960.05

MINT AT NEW ORLEANS.

January	687,500.00	677,259.45	..	..	687,500.00	677,259.45	800,000	122,740 55
February	687,500.00	673,042.26	..	..	687,500.00	673,042.26	800,000	126,957.74
March	816,406.25	794,886.45	..	..	816,406.25	794,886.45	950,000	155,113.55
April	782,031.25	759,862.75	..	..	782,031.25	759,862.75	910,000	150,137.08
May	778,437.50	752,656.92	..	..	778,437.50	752,656.92	900,000	147,343.25
June	794.921.87	772,705 86	16,008.51	15,561.11	810,930.38	788,2'6 97	925,000	152 294.14
July	85,937.50	83,396.61	..	..	85,937.50	83,396.61	100,000	16,603.39
August	515 625.00	499,788.17	394.27	382.12	516,009.27	500,120.29	600,000	100 261.83
September	687,500.00	658,041.04	6,989.94	6,690.42	694,489.94	664,731.46	800,000	141 958.96
October	687,500.00	652,222.57	..	..	687,500.00	652,222 57	800 000	147.777.43
November	687,500.00	647,010.38	..	..	687,500.00	647,010.38	800,000	15 ,081.62
December	687,500.00	643,398.03	..	..	687,500.00	643,398.03	800,000	156,601.97
Total	7,893,859.37	7,614,220.49	23,392.72	22,633.65	7,916,707.54	7,636,854 14	9,185.000	1.570,779.51

The operations of these institutions are confined by law to the receipt, melting assaying, and stamping of bullion, and returning to depositors the identical bullion in unparted bars, or, in the case of gold bullion, the value in coin or unparted bars, as may be preferred by the depositor.

1. For melting, refining by fluxes, assaying and stamping of bullion, ½ of 1 per cent. on the gross value of the bullion.

2. When more than one melt is necessary, or extra materials are required to remove the base metals contained in deposits to render the resulting bullion ductile, an extra charge corresponding therewith must be made.

3. Upon gold bullion deposited for payment of its value from the bullion fund, in addition to the assay office charge, the same charges as those imposed by the mint to which such bullion is required to be forwarded, and a further charge to cover the cost of transportation, will be collected of the depositor.

The Coiner of the Mint at San Francisco* operated on 2,848,812.810 standard ounces of gold bullion during the year 1886, and returned in settlement an excess of 124.654 standard ounces. The same officer operated on 65,709.70 standard ounces of silver, and delivered in settlement an excess, over and above the amount charged to him during the year, of 43.79 standard ounces

The following statement exhibits the work of the Coiner's department of this mint from 1882-6.

COMPARATIVE STATEMENT OF OPERATIONS IN THE COINER'S DEPARTMENT OF THE MINT AT SAN FRANCISCO FOR THE FOUR YEARS ENDED JUNE 30, 1886.

Gold Coinage.

Fiscal Years.	Amount.	Per cent.	Legal Allowance.	Actual waste.	Per cent. of allowance.
	$		Stand. ozs.	Stand.ozs.	
1882-83 . . .	26,760,000	54.7	1,325.685	134.715	10.1
1883-84 . . .	23,543,500	54.3	1,164.590	120.300	10.3
1884-85 . . .	20,857,500	54.7	1,041.746	70.337	6.75
1885-86 . . .	27,080,000	52	1,424.406	47.018	3.3
Total and mean .	98,241,000	...	4,956.427	372.370	7.5

* Report of the Director of the U. S. Mint for 1886, p. 27.

Silver Coinage.

	$				
1882-83 . . .	7,350,000 00	51.5	12,628.130	638.76	5
1883-84 . . .	5,850,000 00	52.5	9,779.060	618.13	6.3
1884-85 . . .	2,908,799 70	53.3	4,703.610	192	4.08
1885-86 . . .	49,066 20	52.8	50.154	.58	1.15
Total and mean	$16,157,865.90	...	27,160.954	1,449.47	5.35

Actual loss for four years last past:

	$
Gold	6,925.70
Silver	1,305.00
	$8,230.70

Deduct for pro rata of proceeds of carpets, used seven years and destroyed:

	$	
Gold value	1,705.60	
Silver value	13.80	
		1,719.40
Net loss for the four years last, coin value		$6,511.30
Legal allowance on gold operated on		87,544.55
Legal allowance on silver operated on		24,844.45
Total allowance for four years past		$112,390.00

It is unusual for a Coiner to return an excess of bullion. This excess arose from gold and silver recovered from old carpets in the adjusting room, which, being unfitted for further use, were burned and produced 171.672 standard ounces of gold and 44.37 standard ounces of silver. These carpets had been in use seven years. The value of the blanks handled in the room covered with these carpets during the period they were down, was, gold, $183,734,000; silver, $46,534866.

STATEMENT OF EXPENDITURES AT THE UNITED STATES MINT AT SAN FRANCISCO FOR THE FISCAL YEAR ENDED JUNE 30, 1886.*

Expenditures.	General Department.		Assayer's Department.	Coiner's Department.	Melter and Refiner's Department.	
	Proper.	Mechanical.			Proper.	Refinery.
	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.
Acids	..	..	1,170.90	43.50	10.80	20,203.56
Charcoal	..	..	694.47	..	589.14	261.84
Chemicals	371.41	13 65	159.97	445.98	1,413.06	206.68
Coal	5,349.13	12 22	..	..	..	1,718.38
Coke	..	..	..	..	1,155.81	
Crucibles, covers, stirrers, and dippers	..	..	21.00	..	1,929.24	26 30
Dry goods	159 90	..	20.26	107.53	251.40	622.25
Gas	1,475.40	..	..	..	..	292.16
Gloves and gauntlets	132.00	..	12.00	405.50	165.00	..
Hardware	223.81	165.72	35.50	41.02	25.77	114.79
Ice	704.53	..	..	..	..	..
Iron and steel	..	25.15	..	18.40	140.91	33.79
Labor and repairs	190.00 / 3,502.23	416.62	51.19 / 939.51	712.81	663.74	192.00 / 2,406.27
Lumber	..	213.67	..	22.38	..	316.66
Machinery and appliances	1,400.00	35.10				
Metal work and castings	69.25	41.47	..	277.00	383.93	126.85
Oil and belting	212.06	427.20	..	94.05	70.38	32.81
Salt	..	..	..	..	..	163.77
Stationery, printing, and binding	194.78 / 406.63	..	54.75	20.50		
Sundries	2,621.96	2.75	5.60	70.60	332.80	78.54
Wood	496.40	..	..	1,795.00	71.25	346.80
Zinc	..	..	..	..	..	1,769.44
Loss on sweeps	..	..	..	..	357.09	344.62
Mittens and sleeves	..	..	..	61.80	1,182.06	
Total	17,559.54	1,353.55	3,165.15	4,116.07	8,392.39	29,257.51
Salaries	25,300.00	..	6,600.00	5,000.00	5 000.00	
Wages of workmen	63,247.85	..	21,080.50	67,908.24	22,418.00	20,405.25
Aggregate	106,107.39	1,353.55	30,845.65	77,024.31	36,310.39	49,662.76

SUMMARY.

Expenditures.	Amount.	Expenditures.	Amount.
	Dols.		Dols.
Acids	21,428.76	Machinery and appliances	1,435.10
Charcoal	1,545.43	Metal work and castings	896.50
Chemicals	2,610.75	Oil and belting	336.50
Coal	7,129.73	Salt	163.77
Coke	1,155.83	Stationery, printing, and binding	194.78 / 481.88
Crucibles, covers, stirrers, and dippers	1,976.55	Sundries	3,162.25
Dry goods	1,261.84	Wood	2,709.45
Gas	1,767.56	Zinc	1,769.44
Gloves and gauntlets	714 50	Loss on sweeps	701.71
Hardware	606.61	Mittens and sleeves	1,243.86
Ice	704.53		
Iron and steel	218.25	Total	64,344.21
Labour and repairs	433.19 / 8,641.23	Salaries	41,900.00
Lumber	552.71	Wages of workmen	204,059.84
		Aggregate	310,304.05

REMARKS.—"Assayer's materials" include matrasses, pipettes, dishes, &c. "Dry goods" include cost of materials for mittens, sleeves, towelling, coin sacks, scale covers, &c. "Labour and repairs" include only temporary labour on repairs. "Sundries" include such articles as cannot readily be classified.

Coinage for the period: Gold, 27,080,000 dols.; silver, 49,066.20 dols.; total, 27,129,066.20 dols. Bars manufactured—silver, 1,845,970.72 dols.

* Report of the Director of the U S. Mint for 1886, p. 36.

Statement of Expenditures at the United States Mint at New Orleans for the Fiscal Year ended June 30, 1886.

Expenditures.	General Department	Refinery.	Total.
	Dols.	Dols.	Dols.
Acids	746.46	491.78	1,238.24
Charcoal	363.90	..	363.90
Chemicals	455.94	..	455.94
Coal	4,253.47	..	4,253.47
Coke	1,459.20	..	1,459.20
Copper	4,538.69	14.40	4,553.09
Crucibles, covers, stirrers, and dippers	4,044.15	..	4,044.15
Dry goods	552.72	54.00	606.72
Gas	2,556.04	..	2,556.04
Gloves and gauntlets	151.70	..	151.70
Hardware	521.24	2.50	523.74
Ice	1,220.50	..	1,220.50
Iron and steel	604.92	44.96	649.88
Labour and repairs:			
Repairs	4,919.95	575.10	5,495.05
Labour	1,100.02	..	1,100.02
Lumber	855.17	..	855.17
Machinery and appliances	1,050.00	..	1,050.00
Metal work and castings	818.93	74.40	893.33
Oils and beltings	1,093.20	..	1,093.20
Salt	3.00	10.40	13.40
Stationery, printing, and binding	240.90	..	240 90
Sundries	4,051.03	21.85	4,072.88
Wood	3,468.15	88.55	3,556.70
Zinc	..	31.42	31.42
Mittens and sleeves	705.53	2.00	707.53
Total	39,774 81	1,411.36	41,186.17
Salaries	31,862.32	..	31,862.32
Wages of workmen	103,996.06	..	103,996.06
Aggregate	175,633.19	1,411.36	177,044.55

Remarks.—"Dry goods" include cost of material for mittens, sleeves, towelling, coin sacks, scale covers, &c. "Labour and repairs" include only temporary labour on repairs. "Sundries" include such articles as cannot readily be classified.

Coinage for the period—silver, 9,300,000 dols.

Statement of Expenditures at the United States Mint at Carson City, Nevada, for the Fiscal Year ended June 30, 1886.

	Dols.	Dols.	Dols.
Gas	375.60	..	375.60
Hardware	36.55	208.00	244.55
Ice	35.91	..	35.91
Labour and repairs	229.00	356.00	585.00
Lumber	50.50	..	50.50
Oils and belting	12.50	..	12.50
Stationery, printing, and binding	8.00	..	8.00
Sundries	887.05	20 31	907.36
Loss on sale of sweeps	355.48	355.49	710.97
Total	1,990.59	939.80	2,930.39
Salaries	10,345.12	..	10,345.12
Wages of workmen	9,085.50	..	9,085.50
Aggregate	21,42.121	939.80	22,361.01

Bars manufactured—gold, 4,293.06 dols.; silver, 810.55 dols.

STATEMENT OF EXPENDITURE AT THE UNITED STATES MINT AT PHILADELPHIA FOR THE FISCAL YEAR ENDED JUNE 30, 1886.*

Expenditures.	General Department.	Engraver's Department.	Assayer's Department.	Coiner's Department.	Melter and Refiner's Department.	
					Proper.	Refinery.
	dols.	dols.	dols.	dols.	dols.	dols.
Acids	..	..	320.96	..	..	3379.87
Charcoal	..	190.04	..	..	2335.21	
Chemicals	58.45	..	58.76	227.46	6085.72	30.75
Coal	12,886.56	..	194.00	..	2126.70	
Copper	..	..	..	..	10,653.13	
Crucibles, covers, stirrers, and dippers	..	..	27.50	..	3577.50	
Dry goods	857.16	..	..	203.46	17.80	35.27
Gas	2,636.32					
Gloves and gauntlets	10.50	..	..	1,964.61	1304.00	
Hardware	425.29	25	60	222.70	131.10	
Ice	645.00					
Iron and steel	107.93	59.09	..	13.63		
Lumber	787.54	..	..	186.59		
Machinery and appliances	87.31	..	..	..	31.14	
Metal work and castings	82.87	..	13.5	3.12		
Oils and belting	107.10	..	..	778.73		
Repairs	465.26	..	..	247.71	473.23	8.00
Salt	..	..	36	..	..	65.55
Stationery, printing, and binding	123.00	..	..	12.75	15.25	
Sundries	9870.77	12.16	175.65	1099.19	1374.99	316.84
Wood	106.42	..	..	4835.04	..	19.36
Zinc	..	..	..	..	..	384.12
Mittens and sleeves	..	..	..	236.00	1042.75	
Total	28,753.48	261.54	783.18	10,080.99	29,168.52	4239.76
Salaries	88,782.89					
Wages of workmen	373,759.73	..	..	..	..	1829.00
Aggregate	441,296.10	261.54	783.18	10,080.99	29,168.52	†6068.76

* Report of the Director of the U.S. Mint for 1886, p. 36.

† Includes 8 dols. due in 1885 for wages paid in 1886.

SUMMARY.

Expenditures.	Amount.	Expenditures.	Amount.
	dols.		dols.
Acids	3700.83	Metal work and castings	87.34
Charcoal	2525.25	Oils and belting	885.83
Chemicals	6461.14	Repairs	1194.20
Coal	15,207.26	Salt	65.91
Copper	10,653.13	Stationery, printing, and binding	151.00
Crucibles, covers, stirrers, and dippers	3605.00	Sundries	12,353.60
Dry goods	1113.69	Wood	4960.82
Gas	2636.32	Zinc	384.12
Gloves and gauntlets	3279.11	Mittens and sleeves	1328.75
Hardware	779.94	Total	73,287.47
Ice	645.00	Salaries	88,782.89
Iron and steel	180.65	Wages of workmen	375,588.73
Lumber	974.13		
Machinery and appliances	114.45	Aggregate	487,659.09

REMARKS—"Assayer's materials" include matrasses, pipettes, dishes, &c. "Dry goods" include cost of materials for mittens, sleeves, towelling, coin sacks, scale covers, &c. "Sundries" include such articles as cannot readily be classified.

Coinage for the period: Gold, 6,997,290 dols.; silver, 20,745,231.75 dols.; minor coins, 17,377.65 dols.; total, 27,660,039.40 dols. Bars manufactured: Gold, 529,017.64 dols.; silver, 26,825.06 dols.

COINAGE EXECUTED AT THE MINTS OF THE UNITED STATES DURING THE CALENDAR YEAR ENDED DECEMBER 31, 1885*.

Denomination.	Philadelphia.		San Francisco.		Carson.†		New Orleans.		Total.	
	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.
Gold.		dols.		dols.		dols.		dols.		dols.
Double eagles	828	16,560.00	683 500	13,670,000.00	9,450	189,000.00	..	..	693,778	13,875,560.00
Eagles	253,527	2 535,270.00	228,000	2,280,000.00	..	..	..	..	481,527	4,815,270.00
Half eagles	601,506	3,007,530.00	1,211,500	6,057,500.00	..	..	..	..	1,813,006	9,065,030.00
Three dollars	910	2,730.00	..	..	..	..	..	..	910	2,730.00
Quarter eagles	887	2,217 50	..	..	..	..	..	..	887	2,217.50
Dollars	12,205	12,205.00	..	..	..	..	..	..	12,205	12,205.00
Total gold	869,863	5,576,512.50	2,123,000	22,007,500.00	9,450	189.000.00	..	..	3,002,313	27,773,012.50
Silver.										
Dollars	‡17,787,767	‡17,787,767.00	1,497,000	1,497,000.00	228,000	228,000.00	9,185,000	9,185,000.00	‡28,697,767	‡28,697,767.00
Half dollars	6,130	3,065.00	..	..	..	..	..	..	9,130	3,065 00
Quarter dollars	14,530	3,632.50	..	..	..	..	..	..	14,530	3,632.50
Dimes	2,533,427	253,342.70	43,690	4,3 9 00	..	..	..	..	2,577,117	257,711.70
Total silver	20,341,854	18,047,807.20	1,540,690	1,501,369.00	228,000	228 000.00	9 185,000	9,185,000.00	31,296,544	28,962,176.20
Minor.										
Five cents	1,476 490	73 824.50	..	..	..	..	..	..	1,476,490	73,824.50
Three cents	4 790	143.70	..	..	..	..	..	..	4,790	143.70
One cent	11,765,384	117,653.84	..	..	..	..	..	..	11,765,384	117,653.84
Total minor	13,246,664	191,622.04	..	..	..	..	..	..	13,246,664	191,622.04
Total coinage	34,458,381	23,815,941.74	3,663,690	23,508,869.00	237,450	417,000.00	9,185,000	9,185,000.00	47,544,521	56,926,810.74

* Report of the Director of the U.S. Mint for 1885-6, p. 89.

† Coinage suspended March 8, 1885.

‡ Includes 1,837 recoined.

COINAGE EXECUTED DURING THE YEAR ENDED JUNE 30, 1886.*

Denomination.	Philadelphia.		San Francisco.		Carson.		New Orleans.		Total.	
	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.	Pieces.	Value.
Gold.		dols.		dols.		dols.		dols.		dols.
Double eagles	84	1,680.00	243,500	4,870,000.00	..	..	..	..	243,584	4,871,680.00
Eagles	348,847	3,488,470.00	694,000	6,940,000.00	..	..	..	..	1,042,847	10,428,470.00
Half eagles	697,629	3,488,145.00	3,054,000	15,270,000.00	..	..	..	..	3,751,629	18,758,145.00
Three dollars	101	303.00	..	..	..	..	..	..	101	303.00
Quarter eagles	4,086	10,215.00	..	..	..	..	..	..	4,086	10,215.00
Dollars	8,567	8,567.00	..	..	..	..	..	..	8,567	8,567.00
Total gold	1,059,314	6,997,380.00	3,991,500	27,080,000.00	..	..	..	..	5,050,814	34,077,380.00
Silver.										
Dollars	20,463,905	20,463,905.00	47,000	47,000.00	28,000	28,000.00	9,300,000	9,300,000.00	29,838,905	29,838,905 00
Half dollars	6,105	3,052 50	..	..	..	..	..	..	6,105	3,052.50
Quarter dollars	14,505	3 626.25	..	..	..	..	..	..	14,505	3,626.25
Dimes	1,746,960	174,696.00	20,662	2,066.20	..	..	..	..	1,767,642	176,764.20
Total silver	22,231,495	20,645,281.75	67,662	49,066.20	28,000	28,000.00	9,300,000	9,300,000.00	31,627,157	30,022,247.95
Minor.										
Five cents	5,519	275.95	..	..	..	..	..	..	5,519	275.95
Three cents	4,519	135.57	..	..	..	..	..	..	4,519	135.57
One cent	1,696,613	16,966.13	..	..	..	..	..	..	1,696,613	16,966.13
Total minor	1,706,651	17,377.65	..	..	..	..	..	..	1,706,651	17,377.65
Total coinage	24,997,460	27,660,039.40	4,059,162	27,129,066.20	28,000	28,000.00	9,300,000	9,300,000.00	38,384,622	64,117,105.60

* Report of the Director of the U.S. Mint for 1885-6, p. 38.

STATEMENT OF BULLION MADE INTO STANDARD SILVER DOLLARS AND WASTAGE AND LOSS ON SALE OF SWEEPS AT THE COINAGE MINTS FROM JULY 1, 1885, TO JUNE 30, 1886.*

MINT AT PHILADELPHIA.

Month.	Manufactured.		Wastage and Sold in Sweeps.		Total Consumption.		Number of Standard Silver Dollars Coined.	Seignorage.
	Standard Ounces.	Cost.	Standard Ounces.	Cost.	Standard Ounces.	Cost.		
1885.		Dols.		Dols.		Dols.		Dols.
July	1,546,875.00	1,499,174.71	1,683.81	1,683.81	1,548,563.81	1,500,863.52	1,800,000	300,826.29
August	1,546,875.00	1,496,185.07	..	..	1,546,875.00	1,496,185.07	1,800,000	304,814.93
September	1,460,980 47	1,397,717.30	2,293.42	2,293.42	1,463,273.89	1,400,010.72	1,700,050	302,332.70
October	1,460,937.50	1,390,920.77	..	..	1,460,937.50	1,390,920.77	1,700,000	309,079.23
November	1,460,937.50	1,382,213.06	..	..	1,460,937.50	1,382,213.06	1,700,000	317,786.94
December	1,461,221.09	1,370,279.67	2,767.46	2,767.46	1,463,988.55	1,373,047.13	1,700,330	330,050.33
1886.								
January	1,460,937.50	1,363,131.16	..	..	1,460,937.50	1,363,131.16	1,700 000	336,868.84
February	1,460,937.50	1,359,941.19	1,984.40	1,984.40	1.462,921.90	1,361,925.59	1,700,000	340,058.81
March	1,461,246.88	1,362,526.66	..	..	1,461,246.88	1,362,526.66	1,700,360	337,833.34
April	1,460,937.50	1,355,724.40	2,495.44	2,495.44	1,463,432 94	1,358,219.84	1,700,000	344,275.60
May	1,460,937.50	1,344,518 14	..	..	1,460,937.50	1,344,518 14	1,700,000	355,481.86
June	1,343,344.92	1,218,388.00	5,958.06	5,958.06	1,349,302.98	1,224,846.06	1,563,165	344,277.00
Total	17,586,168.36	16,540,220.13	17,187.59	17,187.59	17,603,355.95	16,557,407.72	20,463,905	3,923,684.87

* Report of the Director of the U.S. Mint for 1886, p. 108.

STATEMENT OF BULLION, &c., FROM JULY 1, 1885, TO JUNE 30, 1886—*continued.**

MINT AT SAN FRANCISCO.

Month.	Manufactured.		Wastage and Sold in Sweeps.		Total Consumption.		Number of Standard Silver Dollars Coined.	Seignorage.
	Standard Ounces.	Cost.	Standard Ounces.	Cost.	Standard Ounces.	Cost.		
1885.		Dols.		Dols.		Dols.		Dols.
July								
August	40,390.62	39,261.23	..	..	40,390.62	39,261.23	47,000	7738.77
September to November								
December	..	..	744.52†	718.93	744.52	718.93		
1886.								
January to May								
June	..	..	1796.10	1656.12	1796.10	1656.12		
Total	40,390.62	39,261.23	2540.62	2375.05	42,931.24	41,636.23	47,000	7738.77
MINT AT CARSON CITY.								
1885.								
July								
August	24,062.50	23,381.71	..	..	24,062.50	23,381.71	28,000	4618.29
September to November								
December	..	..	1495.78	1453.46	1495.78	1453.46		
1886.								
January to March								
April to June								
Total	24,062.50	23,381.71	1495.78	1453.46	25,558.28	24,835 17	28,000	4618.29

* Report of the Director of the U.S. Mint for 1886, p. 108.

† Includes .01 standard ounce; cost 0.02 dols. transferred to recoinage account.

STATEMENT OF BULLION, &c., FROM JULY 1, 1885, TO JUNE 30, 1886—*continued.**

MINT AT NEW ORLEANS.

Month.	Manufactured.		Wastage and Sold in Sweeps.		Total Consumption.		Number of Standard Silver Dollars Coined.	Seignorage.
	Standard Ounces.	Cost.	Standard Ounces.	Cost.	Standard Ounces.	Cost.		
1885.		Dols.		Dols.		Dols.		Dols.
July	85,937.50	83,396 61	..	..	85,937.50	83,396.61	100,000	16,603.39
August	515.625.00	499,738.17	394 29	382.12	516,019.27	500,120.29	600,100	100,2 1.83
September	687,500.00	658,041 04	6989.04	6690.42	694,489.94	664,731.4	800,000	141,268.96
October	687,500 00	652,222 57	..	..	687,500.00	652,222.57	800,000	147,777.43
November	687,500.00	647.010.38	..	..	687,500.00	647,010.38	800,000	152,989.62
December	687,500.00	643,398.03	..	..	687,500,00	643,398 03	800,000	156,601.97
1886.								
January	773,437.50	721,719 76	2348.17	2191.15	775,785.67	723,910.91	900,000	176,280.24
February	773,437.50	719,839.50	..	..	773,437.50	719 849.50	900,000	180,160.50
March	773,437.50	718.377-03	..	..	773,437.50	718,377.03	900,000	181,622.97
April	773,437.50	716,163.90	..	..	773,437.50	716.163.90	900,000	183,836.10
May	773,437.50	711,237.04	5963.12	5501.95	779,4?'.62	716,738 99	900,000	183,762 93
June	773,437.50	701,046.90	7473.71	6774.19	780,911.21	707,821 09	900,000	196,958.10
Total	7,992,187.50	7,472,190.93	23,189.21	21,589.83	8,015,376.71	7,493,780.76	9,300,000	1,827,809.07

* Report of the Director of the U.S. Mint for 1886, p. 110.

PARTING PROCESS USED IN THE UNITED STATES ASSAY OFFICE, NEW YORK CITY.

The Sulphuric Acid Process.

The United States Assay Office at New York was established by Act of Congress of March 4, 1853.* It was opened for the transaction of business on October 10, 1854. Intended to accommodate the bullion interests centering in New York, it has been extensively used by miners, bankers, brokers, importers, manufacturers, jewellers, &c., and has proved a convenience to the public generally throughout the United States. Any possessor of bullion, the value of the gold and silver in which is $100 or over, can deposit it and receive payment for the gold in coin or fine bars, and for the silver in fine or standard silver bars. These gold and silver bars have their weight and fineness stamped on them, and pass current at that value. They have largely taken the place of gold and silver coin for consumption in various arts and manufactures. The coin, always the best attainable, thus saved from destruction amounts to about $12,000,000 annually.

The office is situated in Wall-street, next to the Sub-Treasury building. It is composed of two buildings. The one fronting on Wall-street is used for the bureaus of the officers, for the weighing and storage of coin and bullion, and the deposit-melting room, which, before the introduction of the gas plant, had five furnaces, Figs. 93 to 111, but now has only two and a gas-melting plant for this purpose. The parting house is in a building in the rear of this. In this building there is a test furnace blown by a piston blower for cupelling the bullion, which sometimes contains a great deal of lead and other impurities. Also for the treatment of the residues and skimmings.

A twenty horse-power engine does the work of crushing the crucibles, débris of furnaces, &c. All the water used in the establishment is furnished by an artesian well in the yard between the two buildings, which 475 ft. deep and 6 in. in diameter. The water is forced up 86 ft. to tanks under the roof. The melting-room for the deposits is a separate department.

* Production of Gold and Silver in the U. S., 1885, p. 275.

Since 1887 gas is being entensively used for melting in many of the United States mints and assay offices. Fig. 117 shows the style of the furnaces which have been very generally adopted. They are made for any size crucible and hold single crucibles from 4 in. up to No. 100 for melting, or a number of small ones for assaying, as high as 10 or 15 being placed in a single furnace at one time. In this last case, as there are no ashes to contend with, and no fear of anything falling into them, they need no cover and can be watched during the whole of the operation.

The furnace consists of an outside of iron, heavily bound and bolted, lined with fire-brick. The combustion chamber is made up of the bottom *a* and the body B, which are securely fastened to the distributing ring C by the hinged bolts O. The burners D are attached to the sides and enter the bottom of the furnace. The supply of gas is regulated by the stop-cock on the gas supply pipe G. The air is brought in at any desired pressure by the pipe F, which also has a stop-cock. It is always placed close to the floor, and would be better if it came up from below the floor, in order to prevent any mixture of gas and air when the blower is accidentally stopped. The parts *a*, B, and C are held up by an iron platform L, and are kept in position with thumbscrews, so that the lining can be easily replaced or any of the parts examined. The cover H swings on the shaft K, and by means of the handle can be raised or lowered, or swung to one side. The feed-hole in this cover is large enough to introduce the charge without removing the cover, which is only swung to one side when the crucible is to be removed. The small cover I, which has an iron handle, is used to cover the feed-hole. In it there is a small observation hole, which is closed with a firebrick plug. The crucible stands on a movable firebrick support in the bottom of the furnace, and in case of an accident there is a drip-pan below to catch the contents. The products of combustion escape through channels in *a*, and also from above, so the heat can be controlled to a certain extent at the bottom, or drawn to the top by opening the cover. The style of the furnace, the crucible used, and the quantity of gas consumed, are given on page 747.

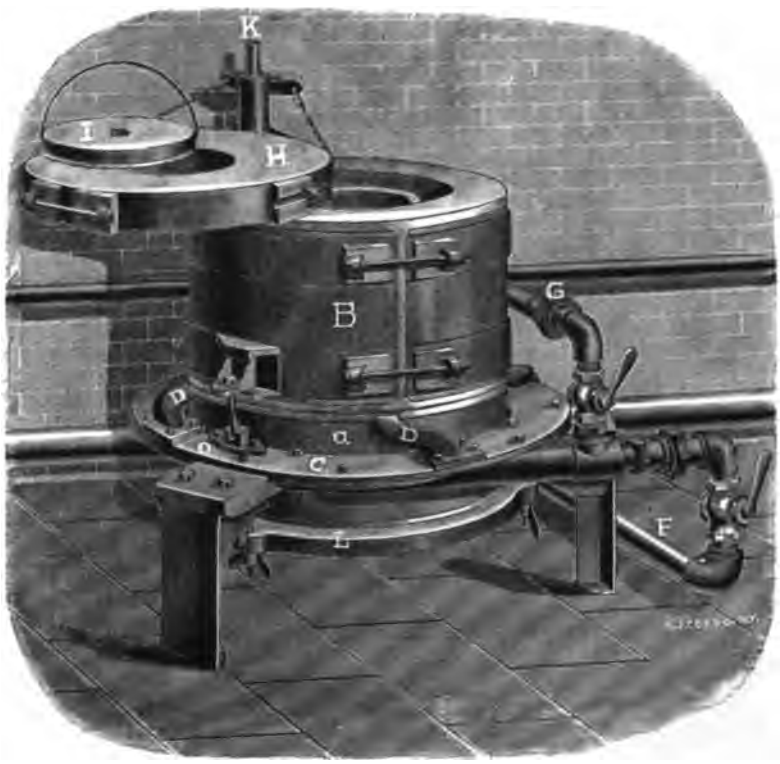


FIG. 117. GAS MELTING FURNACE. *To face page* 746.

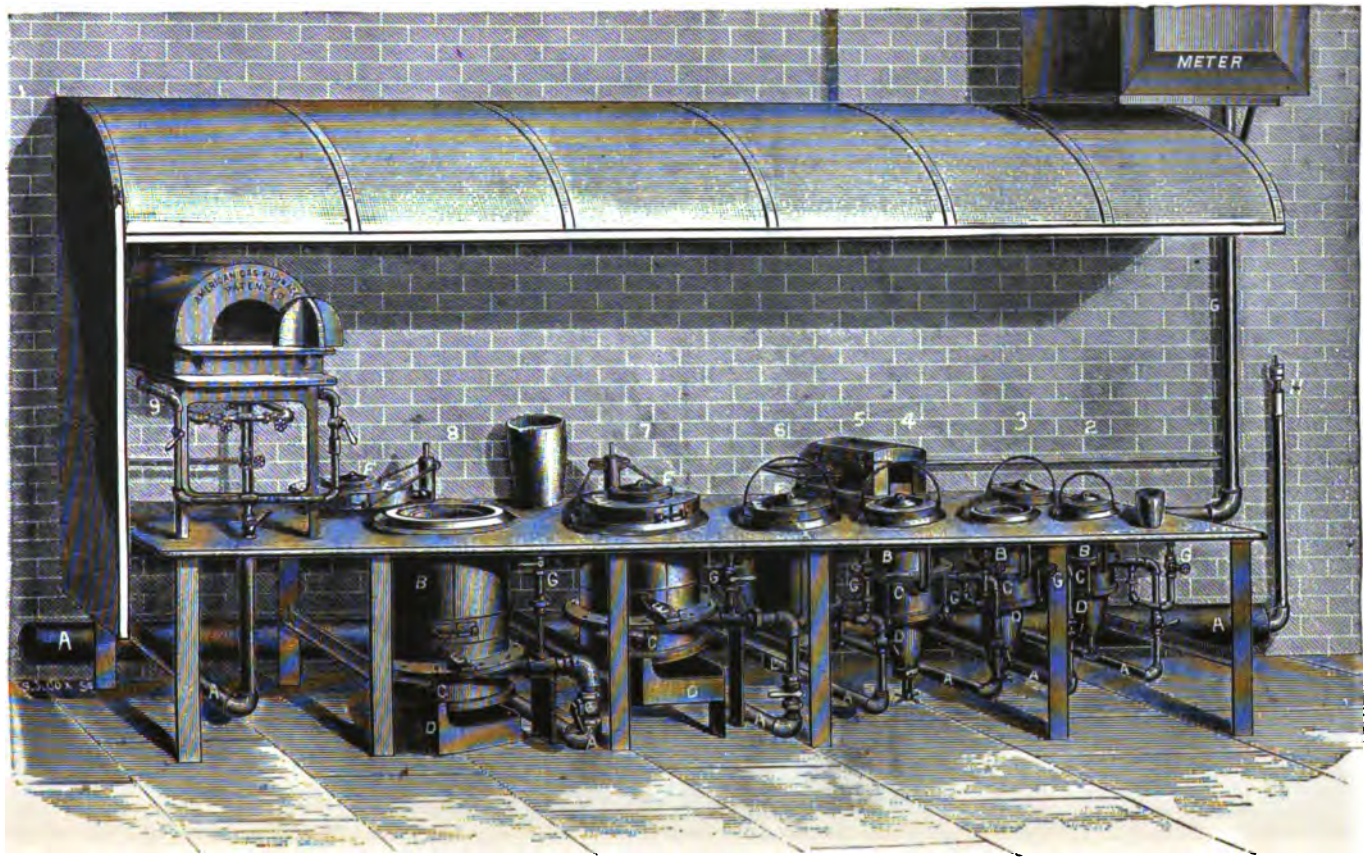


FIG. 118. GAS MELTING PLANT AT THE UNITED STATES ASSAY OFFICE, NEW YORK. *To face page* 747.

Number of Furnace.	Size of Crucible.	Cubic Feet of Gas consumed per Hour.
1	1 in. to 4 in high	15 to 18
2	5 in. high	30 ,, 50
3	8 ,, ,,	75
4	12 ,, ,,	90
5	20 ,, ,,	240
6	25 in. to 40 in. high	350
7	50 ,, 70 ,, ,,	420
8	75 ,, 100 ,, ,,	480

A little larger or smaller crucibles can be used if desired. The same system is applicable to muffle furnaces, and can be adapted to any other metal besides gold and silver. Their great advantage is their cleanliness, freedom from ash, and the readiness with which they can be lighted and put out.

The gas furnace plant of the New York Assay Office, Fig. 118, is on one side of the deposit melting room, and consists of six melting furnaces, 2, 3, 4, 6, 7, and 8, a muffle furnace 9, and a furnace 5 for heating the stirrers. This plant was put in in 1888. The air for the blast is taken from outside the building. The blower is a No. 1 Root, and is hung upon the wall, giving a pressure of 2 lb.; the regulator H having a series of weights which regulate the pressure by the blowing off of the air when it exceeds the pressure required. All of these furnaces are set in a cast-iron platform 14 ft. 6 in. long, and 3 ft. 6 in. high, the lower part being entirely open so as to admit of constant inspection. A is the air pipe with a supply cock, B the body of the furnace, C the bottom of the furnace, D the drip-pan, E the burner, F swing or hand cranes, G the gas-pipe with a supply cock, and H the pressure gauges. There are six of these melting furnaces, whose sizes are given below, in order to admit all weights of melts.

2	corresponds to crucible	2,	takes a	5-in. to 7-in.	blacklead	crucible,	holding	150 to 175 oz.
3	,, ,,	3	,,	10-in.	blacklead	crucible,	holding	300 to 350 oz.
4	,, ,,	4	,,	16-in.	,,	,,	,,	600 oz.
6	,, ,,	5	,,	20-in.	,,	,,	,,	800 oz.
7	,, ,,	6	,,	50-in.	,,	,,	,,	3000 oz.
8	,, ,,	7	,,	80-in.	,,	,,	,,	8000 oz.

No. 5 is a furnace for holding the stirrers, and No. 9 a cupel furnace. All these furnaces have covers, which are removed by a wrought-iron handle, except Nos. 7 and 8, whose covers are so heavy that they are made to swing on a post. Each cover has

a plug for examining the charge while in fusion. The amount of gas consumed by each furnace varies somewhat according to use. For the plant at the New York Assay Office it varies from 24,000 to 26,000 cubic feet per month. All of these furnaces are set under an iron hood to carry off the fumes. They are used exclusively for melting gold deposits, the bricks and bars for which are usually small. For silver deposits two of the old style of furnace are still used, not only because they hold more crucibles, but because being large, a full-sized silver brick can be heated preparatory to melting up to redness on the outside of the pot, while the other brick is being melted in it, and much time is thus saved, but they have no other advantage over the gas plant. It takes three of the gas furnaces to do the work of one of the other style, since the gas furnaces allow of putting smaller crucibles in the corners of the furnace around the larger ones, and also of using a very large crucible. The same kind of hood, Fig. 113, is used in charging the crucibles in the gas furnace as are used in charging the square furnaces, but it is a little different shape. After two years run it is found that there is no appreciable difference between the cost of the fuel in the two kinds of furnaces. There is, however, no dust in the gas furnaces, and there is also no ash, and very much less débris of furnaces to be re-treated for precious metal. The ease and facility with which the furnaces can be lighted is a great advantage, as is also the rapidity with which the heat is got up. In some of the Western assay offices these furnaces are used for all kinds of melts to the exclusion of any others. They will undoubtedly be introduced into the mints.

The deposit melting furnaces require the attendance of four men. When, however, the deposits are received faster than the deposit room furnaces can treat them the furnaces in the parting house are used for melting. The accounts of this department are kept entirely separate from the parting establishment, and whatever bullion goes out of it is charged to the parting house which is called the melting and refining department.

To determine how much coin or bars is to be exchanged for a deposit,* it is melted in a crucible by itself, and poured into iron

* *Ibid.*

moulds. The exact proportion of gold and silver in the resulting bars is ascertained by the assay of small samples. These are taken from the gold bars by cutting. They are taken from silver deposits by dipping from the fluid metal in the crucible after a careful stirring, and granulated in cold water, each granule thus obtained being a true representative of the whole melt. This method was adopted on account of the tendency of silver when alloyed to segregate in cooling. By calculation, the proportion indicated by assay gives the amount of gold and silver in the weight of the deposit after melting. Proper deductions are made for the cost of parting, refining, &c., and the net result is paid to the depositor.

The deposits of gold and silver bullion during the year 1885-6 amounted to $22,559,482.11 against $35,248,421.34 in the previous year; a falling off of nearly $13,000,000. This falling off in deposits of gold was chiefly in foreign gold bullion and coin. The value of the gold bars manufactured during the year was $15,820,585.07, and of silver bars, $6,721,393.36. The melter and refiner operated during the year on 929,770.775 oz. of gold, and returned at settlement an excess of 889.447 oz. of the value of $16,547.85. The same officer operated on 5,920,653.13 oz. of silver bullion, with a wastage of 318.10 oz. The amount parted and refined at the acid refinery was 4,656,786 gross ounces, containing 540,041 standard ounces of gold and 3,831,572 standard ounces of silver. There were received from the acid refinery 540,408 standard ounces of gold and 3,654,582 standard ounces of silver. There was used in the parting operations, 1,729,327 lb. of sulphuric acid. There were sold 2,531,810 lb. of waste acid, and 321,477 lb. of blue vitriol, realising $17,267.86.

The value and classification of the deposits at the Assay Office at New York during the fiscal year ended June 30, 1886, for bars of gold and silver, probably for use in the arts and manufactures, is given in the Table on the next page.*

The process used is known as the double or the sulphuric acid, or the Mason process,† and was invented by Mr. Andrew

* Report of the Director of the U.S. Mint for 1886, p. 60.

† For the history of the invention of this process by Mr. Mason, see "Production of Gold and Silver in the United States," 1886, page 275.

Mason, the present Superintendent of the New York Assay Office, in the year 1866, at which time he was the melter and refiner, and resulted in very greatly reducing the expenses of parting.

Classification of Deposits.	Gold.	Silver.
	$	$
United States coin	...	215.78
Foreign coin	103,720.67	72,761.31
,, bullion	502,872.05	730,728.34
Plate, etc..	747,034.29	191,307.40
Domestic bullion	2,017,296.50	3,641,093.48
Large gold bars exchanged for gold coin, and redeposited for small bars, less the charges and fractions paid in gold coin	1,545,999.14	
	4,916,922.65	
Large gold bars exchanged for coin, and taken by manufacturers . . .	1,664,535.02	
Total	$6,581,457.67	$4,636,106.31

The process consists of eight operations:

1. Inquartating and granulating the bullion.
2. Solution of the granulated bullion in sulphuric acid.
3. Sweetening and fusion of the insoluble residue of gold.
4. Condensation of the acid fumes.
5. Reduction of the sulphate of silver by copper.
6. Sweetening and melting of the precipitated silver.
7. Treatment of the sulphate of copper.
8. Treatment of the débris of furnaces, sweepings, &c.

I. Inquartating and Granulating the Bullion.

When the bullion is turned over to the parting-house they recognise several varieties of it, which are carefully separated so as to treat them in the various mixtures. All the material received is first melted and inquartated so as to have two of silver and one of gold, or three to four of silver to one of gold, the attainment of a high fineness of gold depending on the amount of copper. Two furnaces are kept constantly on granulations, the pots being charged with 160 oz. to each melt. A day's work is eight hours, in which time a man can granulate 1200 oz. or 8 melts; generally

they do not make more than 600oz. Each furnace requires one man. The charge for melting is made so that the copper does not exceed one-twelfth part of the entire weight of the metal. It is generally preferable not to have it above 6 per cent. It is usual to adjust the gold, when good silver is plenty, so that it will contain about 6 per cent. of copper and 75 of silver. In any case the relation of the gold and the copper must be not over one-third and one-twelfth respectively of the entire weight of the bullion. When the copper contains not more than 8 per cent. of silver and no gold, it is melted with the reducing-house scrap, and used to precipitate the silver, or melted with similar silver of a higher fineness, and used in making standard bars. When it contains gold without silver it is used to dilute silver containing too little gold, or kept to make standard gold. If it contains both it is treated like any other deposit.

The gold, especially that which comes from California and the Frazer's River district of British Columbia, often contains a considerable quantity of platinum and the platinum metals. This is a matter of great inconvenience, for when iridosmene is found in the jewellers' bars it makes a great deal of trouble. It is always desirable to separate it, and in order to do it, where it is suspected, the cupels and cornets of the assay are carefully watched for the dark specks. If not detected here, it shows itself in the hard dark spots in the manufactured article. It is always separated when found. This is done by first ascertaining how much platinum and iridosmene are present by an assay. It is then melted with from four to five times its weight of pure silver, so as to make at least three or four parts of silver to one of gold. The more silver the better, as it increases the difference in the density of the metals. When fluid it is left quiet in a liquid condition for half an hour. The platinum metals by their gravity settle to the bottom in the transfer. It is then poured off. As soon as in the transfer the metal becomes thick the pouring is stopped. The crucible is put back into the furnace and the heat brought up so that the material accumulated on the sides will melt. In this way a "king" is formed at the bottom. This is allowed to stand in the fire over night, and is then assayed. If gold is found in it, it is melted again with two to three times its weight in

silver, and the same process repeated. Usually after the second operation it will be found that there is no appreciable quantity of gold in it. The platinum metals will thus be found entirely separate from the gold and in a considerable quantity of silver. This is accumulated in the vaults as metal containing platinum. After a long period of time it is dissolved in sulphuric or nitric acid. The amount is not very large, about 150 oz. only having accumulated in about ten years. The separation is never done until it has accumulated sufficiently to make it worth while.

In the refining house there are seven furnaces of the old type using coal, two of which run constantly on granulations. The crucibles used are of graphite. They will usually last six melts of six to seven hours each; after this they either crack or are worn so thin that it is no longer safe to use them. The charge for a granulation silver melt is always 160 lb. The furnace is the ordinary one with an inclined sliding door, heated with anthracite coal. When the melted charge is nearly ready to granulate, the cover of the crucible is removed, and a small cup of burned fire-clay, made for the purpose, holding about one quart, is put on the top of the melted mass to be brought up to a red heat. When it is ready for granulation this cup is seized by special iron tongs and the metal thrown with a peculiar rotary motion which has for its object to make the metal strike the water in different places and in a flat stream, so that the granules will be thin and flat, and not lumpy or spherical, and thus expose to the acid the greatest possible surface. Slight changes in the manipulation make very decided ones in the result. It requires a considerable amount of dexterity to throw the stream of metal so as to give the granules the desired shape. The vessel into which the metal falls is made of copper, and is 30 in. in diameter and the same in depth. Inside of this, fitting the vessel loosely at the bottom, is a round copper pan 5 in. deep, with three handles on the side, so arranged that when lifted out by a Weston pulley-block attached to a swinging crane it will tip to one side, where there are holes in the bottom. It is raised to the top of the cylinder, set down on it, and left there to drain. When the water ceases to run from it, it is carried by the crane to an inclined cast-iron pan 43 in. long by

27 in. wide and 4 in. deep, having flaring sides. Into the bottom of this pan a piece of gas-pipe is screwed, which conducts all the water into a wooden box lined with copper. The water draining off always carries with it some small pieces of the granules. The granules are left in the iron basin to dry, and are then stored in wooden boxes lined with copper having two iron handles on the outside. The boxes are made of 1-in. pine, 18 in. long, 11½ in. wide, and 6 in. deep. These are piled one on the top of the other until ready for solution. The amount granulated at a time is from 150 oz. to 160 oz. The melting lasts from six to seven hours every day. The granulations are made once an hour, so that each pot lasts about thirty-six heats. The pot used in melting for the granulation is a No. 70 graphite pot. Two or three will generally be used in a week's work. Two men do all the work of granulation. There are two granulation furnaces, which, with the crane and granulating tank, are together. When there is any necessity, however, any one of the other furnaces may be used for this purpose. The granulations when drained are placed in wooden boxes until ready to be dissolved.

In some of the establishments, as at Philadelphia, great care is taken to have the water used for granulation as cold as possible, ice being put into it when it is not sufficiently cool. In others, special care is taken to have the water in constant motion. In others the water is made to flow in continually, the overflow being carried off by a special pipe. In others the water is allowed to remain at the temperature of the air, but is drawn off with a syphon after every operation.

II. Solution of the Granulated Bullion in Sulphuric Acid.

The granulated bullion is treated in a series of ten cast-iron pots. Nine of these are of large size, and one is small. The large-sized pots are placed in two rows of three, one row of two, and one by itself, and the small one at the end of the row of two, all having fireplaces underneath them. They are heated with anthracite coal. The dimensions of the larger pots are 45 in. in diameter, 26 in. deep, with a rim 3 in. wide. They are

1½ in. thick. Each is set in a cast-iron plate 64 in. square and 36 in. from the ground. They weigh 1100 to 1200 lb. each, and hold about 160 gallons of acid. The small pot is 24 in. in diameter and 26 in. deep. It weighs about 850 lb., and holds about 50 gallons of acid. It is set in a cast-iron frame 50 in. by 50 in., and 33 in. above the ground. This small pot is only used for boiling the gold from the silver residues. It is used only because from the construction of the floor there is no place to set a larger one. These furnaces are built with the ashpit below the floor, which makes them just high enough for convenience. All of them are covered with hemispherical hoods, which are made of 8-lb. lead. In order to keep them from sagging they are supported on the inside by eight iron ribs. These hoods connect with the condensing apparatus by a pipe 10 in. in diameter, having a hydraulic seal, so that it can easily be taken down when necessary. When these pipes run horizontally they are supported below by slats of wood held in place by iron straps. Each of the hoods has an opening, 17 in. high by 26 in. long, for the introduction of the alloy and for working in the pot. This is closed by a sheet of lead, in which there is an opening 13 in. long by 5 in. high, which is also closed by a sheet of lead. These sheets are simply set against the pots with a part of the cover turned over at the top, so as to allow of their being easily removed. The smallest opening is used for stirring the charge. The largest one for introducing and withdrawing it. Two of the largest pots are generally used for the solution of the silver granules. The other seven are used on gold. Any one of them may be used on granulations. The charge is 300 lb. to 400 lb., over which from three to four times that weight of acid at 66 deg. B. is put. This is for the ordinary alloy. When the bullion is 850 fine a charge of 400 lb. is made, but when it is only 800 fine the charge is reduced to 350 lb. When the bullion is very base a charge as low as 300 lb. is used. The amount of acid used is generally 4 lb. to the pound of inquartated bullion. A copper charge with only 55 per cent. of silver has been worked directly. In the year 1884, 300,000 oz. to 400,000 oz. of this kind of bullion, with only 55 per cent. silver, 45 per cent. of copper, was treated directly with acid, but

the charge was only 250 lb. The lowest ever treated before was 76 per cent. silver to 24 per cent. copper. The usual charge is about 300 lb. of granules. The charge of acid for gold is made in two successive additions, the first charge being about 650 lb. With silver the additions of acid must be frequently made to keep the charge from boiling over. In some places, when there is very little gold in the silver, the charge is sometimes made much greater, as the amount of gold remaining behind in the pot would be too small to treat. The amount of gold in the New York Assay Office is, however, almost always about the same. If there is any considerable amount of copper in the granules, the charge has to be smaller, as the copper requires more acid than the silver.

The charge once in the pot, the acid is gradually raised to the boiling point and kept there for several hours during which time 250 lb. to 300 lb. of acid is gradually added. When the solution is complete, the silver sulphate is drawn off by means of a syphon and run into vats on the floors below. This is done through a lead box set in the floor, 18 in. by 12 in. size and 5 in. deep, divided into four compartments, each one of which communicates by a pipe with a precipitating vat on the floor below. The syphon from the pot fits the pipe in this compartment tight. When not in use this box is closed by a sheet of lead level with the floor. Considerable care is required in performing this operation in order not to carry over any gold, and to effect this the gold is carefully levelled over the bottom of the boiler, a lead syphon with the end turned up about three inches is then introduced to draw off the supernatant liquor. The fire is then withdrawn, and fresh acid is placed over the gold, which is then ladled out with an iron ladle perforated with holes and placed in a cast-iron scoop, 22 in. long, 14 in. wide on top, 8 in. wide on the bottom, and 10 in. long, which fits into one of the holes in the conical cover of the pot. This scoop, for convenience of handling, has three handles, one on the end and one on each side. Almost every particle of gold is in this way drawn out of the pot. From the scoop it is transferred to the pots on the other side of the room, each one of the pots receiving the charge from one of the pots of the same size directly opposite

to it. This boiling is repeated seven times. After the fifth boiling the gold is washed in a tub first with cold water then with hot. These washings are poured into a trough lined with lead and below the level of the floor. It is 5 ft. long by 18 in. wide and deep. The liquor runs into the tanks on the floor below. Occasionally in removing the gold from the pots some silver sulphate insoluble in the concentrated liquors is thrown down with it. This is put one side to be treated by hot water and acid when it is convenient. It has been found occasionally that some lead sulphate remains in the gold, and as this is insoluble in sulphuric acid, it is likely to make the gold brittle, so that it has to be toughened after the melting. In order to remove the lead in the washing, as the lead sulphate is in very fine particles and lighter than the gold, the vessel in which the gold is washed with hot water is slightly inclined so as to carry off the particles of lead sulphate with the liquid as it runs off. In some of the mints this has succeeded so well that no toughening of the mint fine gold is required. The amount of gold ladled over from one pot to the other will not be far from 100 lb. The gold is boiled with about 200 lb. of acid in these pots for from two and a half to three hours. This acid is then removed, and it is boiled again with a fresh supply of acid. After two boilings in the same boilers the acid is removed and the charge in both of them is put into the third and boiled there twice, and so on; the seven boilings last five days. The gold in this way is made 998 to 999 fine. The finishing acids are syphoned off into a tank and forced into a receiver in the attic to be used once again. The metal with the gold is silver. When the gold was in considerable quantities in the bullion received the first solution was made in nitric acid, and a single subsequent solution was made in sulphuric acid as now, which uniformly gave gold 998 fine. As soon as the quantity of gold became large it was abandoned for the present process. Four men are required for the work of these furnaces. They do all the work of transfer, washing and sweetening. All the men engaged on the acid and crystallisation works are obliged to wear long rubber gloves and boots to protect their hands and feet. With four pots a day, on charges of 300 lb. of granulations, one

ton of precious metals a day can be conveniently treated. A larger amount could be treated but the work would be difficult.

The amount and cost of the acid used in the fiscal year ending June 30, 1882, is given below:

		$
Sulphuric acid, 66 deg. B. ...	1,579,340 lb. at $1.25 =	19,741.73
,, liquor, 40 deg. B. ...	2,108,570 lb. at 25 cts. =	5,271.17
Net cost		... 14,470.56
Less 200 brls. blue vitrol exchanged		... 4,266.11
		$10,204.45

III.—Sweetening and Fusion of the Insoluble Gold Residue.

After the gold has been boiled it is collected in the three-handled iron scoop, emptied into a lead-lined tub where it is washed several times by decantation, and then transferred to a wooden vat upon wheels, lined with 6-lb. lead, where the sweetening is completed. This tank is 28 in. by 33 in. and 17 in. deep. A wooden filter filled with holes is placed upon its bottom, the top of which is 4 in. above the bottom. The bottom is made to incline towards the centre, and from the lowest point there is a pipe which is made to communicate with the reservoir into which the liquid is to run. The wooden filter is carefully covered with a cloth, and the gold is put upon this cloth and washed once with cold and twice with hot water until it is entirely sweet. This takes from two to three hours. When the gold is quite sweet it is melted. As the gold which comes from the sulphuric acid process is generally quite coarse, it can be fed into the melting pots without previous pressing, for when the charging hood is placed over the pot there is little danger of loss, and by this process the gold is so coarse that there is little fear of its being mechanically carried off. If it is very fine it is pressed into cakes by a hydraulic press. These cakes are 12 in. in diameter and 3 in. thick, and weigh from 130 lb. to 150 lb. The cakes of gold are dried in an oven heated with steam. They are then broken up and melted in graphite crucibles and fluxed with a cover of bone-ash into which nitre is introduced as in the silver melting. The

material remaining in the furnace from three to five hours is then cast into bars which run from 997.5 to 999 fine.

The charge for a gold bar melt is 6000 oz., a No. 80 graphite crucible being used. The charge for a silver bar melt is usually 5000 oz., a No. 100 crucible being used. The casting is done in the same way as the silver. The moulds are of iron with a cover, and are made of various sizes to suit the demand for bricks of various values. Those for bricks which will have a value of about $8000 are 7 in. by 3½ in. by 1¾ in. Those worth about $5000 are 6 in. by 2¾ in. by 1⅜ in. Those which are 2¾ in. by 1½ in. by 1 in. hold about 32 oz.; the brick is worth $640 to $650. The size 2¼ in. by 1½ in. by 1 in. holds 20 oz.; the brick is worth above $400. The size 2 in. by 1¼ in. by ½ in. holds about 15 oz., and the size 1¾ in. by 1 in. by ½ in. holds 10 oz. These bricks are all stamped with the stamp of the office, and pass current in exchanges. Only one furnace is specially designated for the melting of the gold. Any other one might be used and often is when the work presses.

IV. Condensation of the Acid Fumes.

The acid from all the extra boilings is collected in two cast-iron pots of the same size as the large solution pots on the same floor; but on a lower level than the bottom of the boiling pots. The acid is syphoned out into an iron launder and runs into these pots. The top of these pots is covered with sheet lead so as to prevent the escape of any acid fumes. The two are connected by means of a curved pipe, and from the centre of the second one a pipe 10 in. in diameter, of 8-lb. lead, goes to the condensing apparatus. The acid which is used is brought to the establishment in large iron tanks, from which it is discharged into an acid egg in the cellar. From there it is forced into tanks in the attic, where are also the water supply tanks for the entire establishment. The acid is conducted in pipes and the quantity which is required is added by means of stop-cocks, it not being necessary to touch the acid at any time. All the acid fumes were formerly carried first into lead chambers which were made of 10-lb. lead. These chambers were 5 ft. by 10 ft. and 7 ft. in height, and were supported on the outside by a wooden frame. About the middle

of this chamber ¾-in. iron pipes covered with lead about an inch and a half apart were placed. These pipes were supported on pieces of wooden joists also covered with lead, and on top of these 30 in. of the best Connellsville coke was placed, the coke being supplied with a constant stream of water. From the top of the vat the gases passed through four vertical pipes 16 in. in diameter, made of 10-lb. lead, in which there was a spray of water from a rose. From here they passed to a central circular chamber, also made of lead and supported with wood, which was 8 ft. 6 in. high by 6 ft. in diameter. This had a coke packing in the centre, and from this they passed through a series of 20-in. pipes 22 ft. high, in which there was a constant series of water sprays. Generally this condensation was quite sufficient for the purpose. There were a double set of these condensers, but when for any reason the condensation was not sufficient the nitric peroxide formed by boiling saltpetre in sulphuric acid was added to the first chamber to collect and condense the gases.

This system, which was adopted for a much smaller plant, was not at times sufficient for the increased work, for which reason, in 1888, a new system was adopted. Since the completion of the new system complaints of annoyance from the fumes, previously not infrequent, have ceased,* and the new apparatus is successfully doing the work for which it was erected. The fumes commence from under the lead hood over the kettles in which the solution of the metals in sulphuric acid takes place. This hood is a dome, not funnel-shaped, as formerly, 2 ft. high and 2 ft. 9 in. in greatest diameter. It forms nearly a circle with the pot, and is believed to have an advantage in its shape from the impinging of the ascending fumes upon the top of the dome causing a certain amount of condensation, thus lessening the quantity to be subsequently treated, as well as the amount of acid to be used. Evidence of this is found in the fine rain of condensed acid frequently seen descending on the metal in the kettle. The hood is connected by a lead pipe 10 in. in diameter and of the required length, with an alternating series of coke chambers made of lead and large vertical and horizontal lead pipes, and with an alkali box, and finally, at the point of exit,

* Report of the Director of the U.S. Mint for 1886, p. 32.

with the main stack. There are in all five coke chambers, about 10 ft. by 5 ft. 6½ in., in which the coke occupies about one-third the cubic space. There are eight vertical lead pipes, 16 in. in diameter by 24 ft. long, six vertical lead pipes 20 in. in diameter by 24 ft. long, and three horizontal lead pipes 20 in. by an average length of 30 ft. The alkali box is 25 ft. long by 3½ ft. high by 22 in. wide, and is so connected and operated that a saturated alkali solution is drawn out by an acid pump and forced up and over the box and rained down at successive points upon the fumes, already largely deprived of their acid. The chambers and pipes are each constantly and liberally sprayed with water, thus keeping the coke and all condensing surfaces cool, and supplying an absorbing element for the gases. The length of the entire apparatus, in a continuous line, would be over 500 ft., and with the stack over 600 ft. The course of the fumes is as follows: From the dissolving kettles into the first coke chamber, thence into the second coke chamber, then into the series of 16-in. vertical pipes, up and down, and into the third coke chamber, on leaving which a horizontal 20-in. lead pipe 20 ft. to 25 ft. long conducts them to the next coke chamber, and so on, the idea being to alternate between the chambers and pipes. From the last coke chamber of the series the fumes pass into the alkali box and are drenched as described previously with alkali solution, passing from thence into the series of 20-in. vertical and horizontal pipes, where they are still further sprayed with water, and are then discharged into the stack to mix with the waste steam from the reducing and condensing houses. Careful inspection at the point of exit fails to detect anything in the gases passing off that can be a source of annoyance.

There was used in the refinery in the year 1885-6 over 1,700,000 lb. of sulphuric acid, a daily average of nearly 6000 lb. At the maximum of work the solution pots are charged with 2000 lb. of metal each day, and all the finishing pots are also in operation. This gives an indication of the large quantity of fumes which the apparatus described is taking care of, and so effectively that complaints have apparently become a thing of the past. A sufficient supply of water is an

indispensable element in the successful working of the system. The artesian well on the premises adequately and economically meets all necessities in that direction.

V. Reduction of the Silver Sulphate by Copper.

All the water used in sweetening the gold is collected in two tanks on the floor below, which are called wash tanks. They are at the end of the room above the level of the precipitation vats. A small amount of residues containing gold and silver are collected from them. They are 7 ft. long, 5 ft. wide on the top, 4 ft. wide on the bottom, and 3 ft. 10 in. deep. They are lined with 8-lb. lead. The bottom is not made level, and their contents can be discharged into the reducers into which the acid silver sulphate runs from the solution pots. There are four of these reducers, which are 14 ft. long, 7 ft. wide, and 30 in. deep. They connect with the stack by a lead pipe 10 in. in diameter. The top of each is covered with two inclined roofs which rise 14 in. above the level of the sides. These roofs are counterpoised in halves so that they lift easily. Between them there is a flat space 16 in. wide, through holes in which the steam pipe and the pipe for carrying off the fumes fit loosely. These tanks are all heated by live steam. It is discharged directly into them by a vertical lead pipe 1½ in. in diameter, which goes to within 2 in. to 3 in. of the bottom of the tank. They were formerly heated by coils, but a burst in the pipe before the entrance of the steam into coils, showed that the liquor could be heated just as well with live steam, and since that time it has been used directly. There is a stopcock on the pipe for regulating the amount of steam.

Before the introduction of the acid liquors, copper plates 10 in. by 21 in. and 1 in. thick are stood around all the sides, leaning against them, and are laid flat upon the bottom; 1000 lb. of copper being charged in each vat. The arrangement described for the precipitation of the silver at the Boston and Colorado Works* would be a much better plan. The object of the plates is to reduce the silver. The acid silver sulphate liquor from the pots in the story above being allowed then to flow into the tank,

* Vol. i., p. 137.

is diluted by the wash water to about 20 deg. B. When the tank is full it is brought up to violent ebullition with steam and kept so for six or seven hours. The silver is precipitated by the copper, and is removed from the tank every other day, two tanks being emptied each day. The copper of the plates is transformed into copper sulphate. They are never taken out until they become too thin or too small to be of any further service. During the precipitation the liquor is constantly stirred with wooden stirrers. The liquor is first tested with a solution of salt. When the silver is all down it is allowed to settle over night. In the morning it will have completely settled. It is again tested to see whether any silver is present. It sometimes happens that the liquor takes up silver over night, so that it must not only be tested at night, but again in the morning. The consumption of copper in precipitating the silver is very much reduced by the melting of the silver deposits which do not contain gold and are under 400 fine with the copper used for precipitation. The silver is thus obtained pure without a direct parting, and the copper alloy takes the place of that amount of copper that would otherwise have to be purchased for it. Occasionally there is a leak in this tank, in which case the liquor must all be removed, and then, in order to get the silver from the copper plates, it will generally be necessary to scrape them. Usually, however, this is not necessary. Whatever tools, such as hoes, shovels, &c., are used during the operation, are made of copper in order not to lose silver, as well as to prevent the liquor from becoming impure. It usually requires twenty-four hours to precipitate the silver. The copper lasts about two weeks, and is then renewed, and the scrap copper is used in making the alloy for the coins.

VI. Sweetening and Treatment of the Precipitated Silver.

When the liquor tested shows that there is no longer any silver present the silver precipitated is drawn to one side with copper hoes so as to be as far from the discharge outlet as possible. The contents of the vat is then discharged into a filter made of copper, 36 in. long, 27 in. wide, and 7 in. deep, the bottom of which is filled with nine rows of holes $\frac{3}{8}$ in. in diameter, at intervals of 2 in. There are also two rows of these holes of the same size

on the sides. It has two handles upon the sides, and is covered with a linen cloth. The liquid contents of the vat is discharged into this filter, and from there runs into a tank. Whatever particles of silver are in the liquor are retained by the filter. The silver is sweetened with hot water in a lead-lined tank upon wheels arranged exactly as the gold sweetener is. These tanks were formerly made round, but that shape has now been abandoned, and the rectangular substituted for it. After the silver is entirely sweet, which will be in about two hours, it is carried in these vats to the hydraulic press, where a pressure of 200 tons is placed upon it, and it is made into cakes 1 ft. in diameter and 3 in. thick, which weigh from 65 lb. to 75 lb. The water which is squeezed out of it runs into a trough around the table of the hydraulic press, and from there through a pipe into a linen bag which is securely fastened to its end, and is suspended over a series of six wooden traps about 1 ft. square. This is discharged into a pipe which runs into the cellar; it then flows through four other traps and from there goes to the sewer. In this way all the silver is collected. These cakes from the press are cleaned of any grains of silver attached to them, and are then placed in the copper steam drier, which runs around the room heated as hot as live steam can make them, and are left about eighteen hours to dry. After this they are broken up and melted. For this purpose a No. 100 graphite crucible is used, the charge being from 4500 oz. to 6000 oz.

In melting the broken silver cakes it is always desirable to have the precious metals under a cover, and this cover is made of bone-ash, which is put over it as soon as the metal is melted. In order to refine the metals this cover is parted in the middle, and about once every half-hour a handful of nitre, with some borax and soda-ash, is put upon the metal. When all the nitre has been absorbed in the bone-ash it is removed and a fresh cover of bone-ash is put on and nitre added again, and this is repeated ten or twelve times during the six hours of melting. The cessation of the action of the nitre is recognised by the workmen by a peculiar motion of the whole cover. The object of the nitre is to oxidise out the base metals, and bring the silver up to 998 to 999 fine. In the other meltings only char-

coal is used as a cover, but it is not necessary, and is not always done, on account of the waste of charcoal.

In front of each of the five melting furnaces there is a ledge on which the moulds for the silver bricks are placed. The silver in casting is dipped out of the pots exactly as in granulating; it is poured into iron moulds which have a tightly fitting cover. These moulds are of several sizes; the largest are 12 in. by 6 in., and 3 in. deep. Such moulds will hold about 1000 oz. Other moulds of the same capacity are 3½ in. and 4 in. deep. The next size is 8 in. long, 3½ in. wide, and 2 in. deep; this holds about 220 oz. Smaller moulds, 5 in. by 2½ in., and 1½ in. deep, hold about 80 oz. Those 5 in. by 2½ in. and 1½ in. deep hold 30 oz. to 40 oz. These are the smallest used. As soon as the metal is quiet any scum that may have risen is taken off, and when it is just about to become solid the iron cover is put on. When solid, but still red hot, it is removed from the moulds and set up on end on a cast-iron pan 48 in. long and 30 in. wide, with a rim 1½ in. deep, to cool. From miscellaneous mixtures it is comparatively easy to get both the gold and the silver up to 990 fine, but very difficult to get it up to 999. The base metal in the silver is copper. Seven furnaces do all the work of melting and granulating the fine silver. The time required to part the bullion and deliver it in bars as fine gold and silver is three days.

VII. Treatment of the Copper Sulphate.

The solution from the silver vats containing the copper is run into concentrating tanks, of which there are four, on platforms 9 ft. above the level of the floor of the copper sulphate room. The liquor which runs into these tanks is only at 20 deg. B., and it is run into them to be concentrated. This is done by means of a lead steam-pipe, which has eight turns and is 320 ft. long, on the bottom of the vat. When it has been sufficiently concentrated it is run into a series of ten intermediate cystallisation tanks directly underneath the concentrating tanks. The tanks are 15 ft. long, 7 ft. wide, and 36 in. deep. The liquor is there brought up to from 35 deg. to 40 deg. B. When at that concentration it is allowed to cool, and the copper crystallises out in two or three days. The mother liquor from these vats

is first concentrated up to 40 deg. B., and then run into storage tanks upon the outside of the building, where it is stored until sold. The copper crystals which are removed from these vats are carried in wooden wagons to the floor above and dumped through a hopper into the solution house. This is a tank 14 ft. long, 7 ft. wide, 30 in. deep, with an inclined roof, the rise in the roof being 12 in. Here it is dissolved, and brought up by steam to 35 deg. to 37 deg. B. From here it is carried by a rubber hose to the finishing crystallisation tanks on the same floor, but below the bottom of the solution vats. There are ten of the finishing crystallisation vats. These tanks were formerly 9 ft. long, 5 ft. wide at the top, and 3 ft. 8 in. at the bottom, and 42 in. deep; eight of them are now 7 ft. 6 in. by 51 in. on top, and 40 in. on the bottom, and 42 in. deep. They are covered with 10-lb. lead. Over the top of these vats are pieces of wood 2½ in. by 3 in.; on each piece there are five strips of 6-lb. lead, 30 in. long and 3 in. wide. They are hung on nails, three on one side and two on the other, and fall into the vat, reaching almost to the bottom; on cooling large crystals of copper sulphate are formed on them. Small ones are also formed upon the bottom and sides of the tank. About once a week, or whenever it is convenient for the work of the office, the mother liquor from the crystallisation vats is pumped up into the concentration vats, and the crystals removed, to do which the lead strips are taken to an inclined lead table 10 ft. long, 3½ ft. wide, with a rim of about 6 in., and lined with 10-lb. lead. The crystals are washed on this table with cold water. The largest crystals are always taken from the lead strips; on the sides and bottom the crystals are smaller. When there are any impurities, such as iron sulphate, attached to any of the masses of copper sulphates, these are washed off. The crystals from the strips and sides come off easily, but the bottom ones have often to be scraped off with an ordinary iron scraper. From the tanks these crystals are brought in a wooden car, 30 in. by 24 in., and 26 in. high. This car is provided with a double bottom with an outlet behind, under which a pail is set to catch any drippings. After washing, all the crystals are carried to wooden bins, made with boards some

distance apart on the bottom, where they dry. They are then packed in barrels, which weigh when filled from 360 lb. to 375 lb. From 100 to 150 barrels per month of this copper are made. The total annual production of copper sulphate is about 450,000 lb. No separation is made between the large and small crystals.

There is a great demand for the copper crystals, which have acquired a certain reputation, and are, therefore, easily sold. The proceeds of the sale nearly cover the cost of the acid and copper purchases. Four men working on two different floors do all the work of the reducing house and of the crystallisation of the copper. The work in each department is intermittent, so that while one is waiting they work at the other. The steam is furnished by the New York Steam Heating Company, but there are two boilers of 60 nominal horse-power each, always ready for use. The power is supplied by a 50 horse-power engine. All the transportation from story to story is done by a lift. All the floors, except the melting room, which is covered with iron plates, are covered with 10-lb. lead. The whole building is fire-proof.

The Table on next page, for which I am indebted to Mr. A. Mason, Superintendent of the New York Assay Office, gives the detailed expenses of the materials used in parting and refining in that office for the fiscal year ending June 30, 1882.

Total Expenses for the Year 1881-2.		Sundries for the Year 1881-2.					
		Labour.		Supplies.		Repairs.	
	$		$		$		$
Acid	10,204.45	General	207.03	Two meters	350.00	Office furniture	48.00
Coal	7,727.82	Mason	664.10	Drying-pan, &c.	164.50	Glazing	6.00
Charcoal	951.40			Under boilers	56.53	Wirerope for elevator	93 00
Tinder-wood	272.00			Addition to boilers	895.00	Scale	3.00
Borax	1,375.92			Five tanks	500.00	Cistern	73.22
Nitrate of soda	867.06			Lumber	192.51	Pulley	30.00
Bone-ash	899.46			Brushes	60.00	Speaking tubes	54.13
Pulverised charcoal	138.88			Chisels, &c.	11.55	Safe lock	25.00
Soda-ash	21.91			Copper hoes, &c.	104.40	Copper leader	28.00
Crucibles, &c.	2,609.60			Brooms	12.00	Mill	103.53
Gloves, hose, &c. (rubber)	997.48			Oil and lard	144.75		
Sheeting and blankets	623.78			Cement	11.40		
Belting	55.78			Thread	5.62		
Ironwork and mason	2,052.99			White lead	8.50		
Lead	572.88			Zinc	90.00		
Firebricks	157.61			Cotton waste	12.00		
Gas	1,192.27						
Water	403.50		871.13		2,618.76		$463.88
Ice	72.30						
Barrels	666.00						
Labour carting ashes	836.00						
Sundries	3,953.77						
	36,652.86						

	$
Labour	871.13
Supplies	2,618.76
Repairs	463.88
Total sundries	3,953.77

The following Table* gives the expenditures of the United States Assay Office for the year ending June 30, 1886:

Expenditure.	General Department.	Refinery.	Total.
	$	$	$
Acids	503.84	17,383.13	17,886.97
Assayer's materials			
Belting			
Charcoal	...	1,165.32	1,165.32
Chemicals	722.84	1,392.02	2,114.86
Coal	120.00	5,024.99	5,144.99
Coke			
Copper	...	5,326.91	5,326.91
Crucibles, covers, stirrers, and dippers	41.10	2,109.35	2,150.45
Dry goods	65.12	791.06	856.18
Fluxes			
Freight and drayage			
Gas	913.77	797.40	1,711.17
Gloves and gauntlets	...	552.30	552.30
Hardware	39.08	233.13	272.21
Ice	27.90	20.94	48.84
Iron and steel			
Labour and repairs	450.00	1,812.25	2,262.25
	560.95	4,386.84	4,947.79
Lumber	...	877.60	877.60
Machinery and appliances	56.85	1,718.22	1,775.07
Metal work and castings	153.32	2,549.73	2,703.05
Mittens and sleeves			
Oils and belting			
Salt	5.00	...	5.00
Sewing			
Stationery, printing, and binding	191.56	...	191.56
Sundries	2,766.14	2,993.73	5,759.87
Telegraphing			
Washing			
Wood	...	231.00	231.00
Zinc	...	70.00	70.00
Total	6,617.47	49,435.92	56,053.39
Salaries	38,828.80	...	38,828.80
Wages of workmen	22,909.50	51,061.37	73,970.87
Aggregate	68,355.77	100,497.29	168,853.06

REMARKS.—"Dry goods" include cost of materials for mittens, sleeves, towelling, coin sacks, scale covers, &c. "Labour and repairs" include only temporary labour on repairs. "Sundries" include such articles as cannot readily be classified.

Bars manufactured—gold, $15,820,585.07; silver, $6,721,393.36.

* Report of the Director of the U.S. Mint for 1886, p. 39.

VIII. Treatment of the Debris of Furnaces, Sweepings, &c.

All the worn-out carpets, furniture and woodwork of the buildings, &c., are burned, and their ashes, with the crucibles, which are no longer serviceable, the débris of furnaces, flux covers, the sweepings of the floors and chimneys, and any other material which may contain gold or silver that has accumulated about the works, are treated. This is put through first a small Blake's crusher, and then through a coffee mill grinder. From here it is accumulated in a receiver which connects by pipes with a series of funnels placed upon shelves. There were, originally, four shelves with four funnels in a row. The upper row only now contains four, the three below only two. The ends of the top ones are bent so as to reach the two funnels on the shelf below which receive the overflow of the two above. The others below discharge each into one below it. These funnels have cones 8 in. in diameter, 8 in. high, and a pipe 8 in. long attached to them. Into these funnels iron tumblers 4 in. in diameter by 6 in. high are placed. The material mixed with water is allowed to discharge into the tumblers of these funnels. The overflow from the tumblers above, carrying the fines with it, goes into the three sets of two funnels below, and from there is discharged into a series of four cast-iron pans 9 in. wide, 18 in. long, and 4 in. deep, whose bottom is covered with mercury. From the middle of the upper of these pans there is a discharge pipe 1¼ in. in diameter, from the next below there is a pipe on each side 1 in. in diameter, and so on through the four. Each of the pans has a discharge cock at the side to draw off the mercury which covers the bottom of the pan when the pan is to be cleaned. The overflow pipe is above the level of the mercury, the discharge pipe below it. From the lower pan the material discharges into two barrels, and what settles there is dried in two copper driers, which are hollow tanks surrounded with steam, and are 6 ft. long by 3 ft. 6 in. wide, and 18 in. high. When dried they are barrelled and sold to the sweep smelters. This apparatus is cleaned as soon as eight barrels of the crushed material has passed through it. This will be every week or ten days. The residues will usually contain from 1 to 10 cents per pound. When

the settlement sweeps are made when the accounts are turned over at the end of the year, as the work must be very hurriedly done, these sweeps will sometimes contain 20 cents a pound. All the residues remaining in the tumblers are collected and melted in pots to a king and separated in the usual way. The amalgam after a time is collected and distilled as usual.

The following Table gives the results of treating sweepings in the U.S. mints and the Assay Office in New York:

Losses.*	Philadelphia Mint.	San Francisco Mint.	Carson Mint.	New Orleans Mint.	New York Assay Office.	Total.
	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.
Melter and refiner's gold wastage	338.30	..	1,333.10	..	..	1,671.40
Coiner's gold wastage	282.43	2,238.11	117.31	..	..	2,637.85
Melter and refiner's silver wastage	..	16,360.42	1,333.27	9,135.63	..	27,329.32
Coiner's silver wastage	6,092.32	620.04	472.96	1,777.60	..	8,962.91
Loss on sale of sweeps	3,031.13	5,951.39	..	..	5,290.48	14,273.00
Total	9,744.18	25,169.96	3,756.63	10,913.23	5,290.48	54,874.48
Paid as follows:						
From contingent appropriation	..	2,474.74	828.22	..	..	3,302.96
„ parting and refining appropriation	..	2,255.91	957.86	..	..	3,213.77
From surplus bullion	1,641.03	16,903.72	287.64	253.46	5,290.48	24,376.33
„ silver profit fund	8,103.15	3,585.59	1,682.91	10,659.77	..	23,981.42
Total	9,744.18	25,169.96	3,756.63	10,913.23	5,290.48	54,874.48

* Report of the Director of the U.S. Mint for 1884, p. 47.

The work of the parting and refining department requires the services of twenty-seven men, as follows: One foreman, one plumber, one pressman, one sweep-washer, two engineers, two granulators, two helpers, four reducers, nine melters, and four boiler-men. Since the steam is taken from the street the boiler-men are not required, but the boilers are always kept in readiness to start at a moment's notice. The plant is lighted by electricity, for which they have their own dynamos.

The vaults of the establishment usually contain $35,000,000 in gold bricks of various sizes to supply the demand for them. The amount of silver which is usually on hand is about $500,000, of which one-half is in bricks in the vaults, and the other half is in various stages of separation in the parting house.

STATEMENT* SHOWING THE NUMBER OF OUNCES, STANDARD WEIGHT, OF GOLD AND SILVER BULLION OPERATED UPON IN THE REFINERIES OF THE UNITED STATES MINTS AT PHILADELPHIA, SAN FRANCISCO, AND CARSON, AND THE ASSAY OFFICE AT NEW YORK.

[Sent to Refineries.]

Month.	Mint at Philadelphia.			Mint at San Francisco.		
	Bullion containing Gold.		Bullion free from Gold.	Bullion containing Gold.		Bullion free from Gold.
	Gold, Standard Weight.	Silver, Standard Weight.	Silver, Standard Weight.	Gold, Standard Weight.	Silver, Standard Weight.	Silver, Standard Weight.
January, 1885	11,576.700	15,230.74	18,639.71	25,776.165	232,104.01	17,117.49
February, 1885	10,726.410	7,068.40	25,025.31	25,682.671	189,317.97	112,911.53
March, 1885	11,812.050	7,950.80	32,576.50	33,001.210	190,789.71	12,082.20
April, 1885	10,798.360	11,019.28	22,158.62	30,101.060	115,196.57	13,368.09
May, 1885	12,097.360	10,980.33	37,257.53	33,589.150	146,195.73	8,280.13
June, 1885	11,994.000	10,465.15	19,876.13	23,026.083	64,730.27	12,442.02
July, 1885	12,827.130	26,996.33	11,943.74			
August, 1885	12,514.690	9,740.54	23,896.49	52,584.673	429,055.13	56,266.20
September, 1885	10,042.490	4,065.14	26,387.87	34,903.467	74,998.66	28,105.20
October, 1885	13,407.250	11,326.50	25,794.48	43,506.289	183,037.32	15,809.39
November, 1885	6,389.340	9,224.19	13,541.67	43,899.651	181,504.63	46,092.47
December, 1885	11,762.580	25,980.74	10,222.57	28,899.484	87,478.11	6,258.20
Total	135,928.360	150,048.19	272,319.62	374,469.903	1,844,408.11	328,732.92
			150,048.19			1,844,408.11
Total silver	..	..	422,367.81	..	..	2,173,141.03

Month.	Mint at Carson.			Assay Office at New York.		
	Bullion containing Gold.		Bullion free from Gold.	Bullion containing Gold.		Bullion free from Gold.
	Gold, Standard Weight.	Silver, Standard Weight.	Silver, Standard Weight.	Gold, Standard Weight.	Silver, Standard Weight.	Silver, Standard Weight.
January, 1885	6,035.961	97,857.88	..	44,487.000	501,503.00	15,790.00
February, 1885	3,576.606	123,537.84	..	34,941.000	290,675.00	5,361.00
March, 1885	1,681.107	23,877.53	..	40,652.000	524,107.00	7,272.00
April, 1885	3,915.341	40,195.67	..	35,018.000	426,655.00	24,894.00
May, 1885	..	..	..	71,165.600	305,409.00	29,012.00
June, 1885						
July, 1885	..	..	..	97,175.000	618,581.00	40,998.00
August, 1885	..	..	..	42,573.000	265,581.00	37,264.00
September, 1885	..	..	..	65,952.000	242,997.00	72,881.00
October, 1885	..	..	..	51,970.000	204,588.00	141,160.00
November, 1885	..	..	..	53,865.000	188,286.00	65,658.00
December, 1885	..	..	..	43,999.000	162,328.00	76,365.00
Total	20,209.015	284,968.42	..	581,797.000	3,730,710.00	516,653.00
			284,968.42			3,730,710.00
Total silver	..	..	284,968.42	..	..	4,247,363.00

RECAPITULATION.

Calendar Year 1885.	Bullion containing Gold.		Bullion free from Gold.
	Gold, Standard Weight.	Silver, Standard Weight.	Silver, Standard Weight.
Mint at Philadelphia	135,928.360	150,048.19	272,319.62
,, San Francisco	374,469.903	1,844,408.11	328,732.92
,, Carson	20,209.015	284,968.42	..
Assay Office at New York	581,797.000	3,730,710.00	516,653.00
Total	1,112,404.278	6,010,134.72	1,117,705.54
			6,010,134.72
Total silver	..	..	7,127,840.26

* Report of the Director of the U. S. Mint for 1886, p. 219.

NUMBER OF OFFICERS, CLERKS, AND EMPLOYES OF THE MINT SERVICE ON NOVEMBER 1, 1886.

Office of the Director of the Mint.

Director.
Examiner.
Computer.
Assayer.
Adjuster.
2 Clerks, class three.
2 ,, ,, ,,
2 ,, ,, two (translator).
Stenographer.
2 Clerks, class one.
2 ,, ,,
Translator.
2 Clerks.
2 ,,
Copyist.
Assistant in laboratory.
Messenger.
Helper.

Mint at Philadelphia.

Superintendent.
Assayer.
Melter and refiner.
Coiner.
Engraver.
Cashier.
Chief clerk.
Assistant assayer.
,, melter and refiner.
,, coiner.
Bookkeeper.
Weigh clerk.
Abstract ,,
Cashier's ,,
Warrant ,,
Register of deposits.
Assistant weigh clerk.
Assayer's computing clerk.
Medal clerk.
Assistant bookkeeper.
Register of warrants.
Assistant.

Assistants, Workmen, and Adjusters.

General Department.

2 Clerks.
6 Assistants.
6 Conductors.
6 Cabinet employés.
16 Night watchmen.
6 Watchmen (post-office building).
6 Carpenters.
3 Engineers.
4 Firemen.
1 Doorkeeper.
1 Messenger.
1 Painter.
1 Gasfitter.
1 Gatekeeper.
1 Millwright.
1 Oiler.
8 Machinists.
2 Adjusters of scales.
3 Blacksmiths.
26 Labourers.
11 Female helpers.
4 Minor coin counters.

Assayer's Department.

1 Assistant.
1 Weigher.
2 Foremen.
2 Workmen.
1 Helper.

Melter and Refiner's Department

1 Office clerk.
1 Foreman.
7 Melters.
5 Deposit melters.
1 Refiner.
2 Sweep-cellar operatives.
1 Furnace builder.
1 Hoister.
1 Gatekeeper.
1 Labourer.
49 Helpers.

Coiner's Department.

1 Chief weigher.
1 Assistant weigher.
1 Hoister.
1 General foreman.
8 Counters (4 ladies).
28 Coining-room employés (21 ladies).
11 Annealers.
14 Cleaners.
27 Rollers.
19 Cutters.
3 Medal-room employés.
93 Adjusters.
48 Temporary adjusters.

Engraver's Department.

2 Engravers.
5 Die makers.

Mint at San Francisco.

Superintendent.
Assayer.
Melter and refiner.
Coiner.
Chief clerk.
Cashier.
Bookkeeper.
Weigh clerk.
Warrant ,,
Abstract ,,
Assistant assayer.
,, melter and refiner.
,, coiner.
Register of deposits.
Cashier's clerk.
Computing ,,
Assistant weigh clerk.
Assayer's computing clerk.

General Department.

4 Clerks.
2 Engineers.
1 Adjuster of weights.
19 Watchmen.
1 Carpenter.
1 Storekeeper.
1 Blacksmith.
2 Machinists.
1 Porter.
2 Firemen.
3 Workmen.
1 Custodian cabinet.
2 Messengers.
2 Doorkeepers.
3 Conductors.
1 Cleaner.
4 Janitors.
1 Helper.

Assayer's Department.

1 Second assistant assayer.
1 Foreman.
1 Weigher.
2 Cupellers.
1 Dissolver.
1 Gold sampler.
4 Weighers.
1 Fireman.

Melter and Refiner's Department.

4 Clerks.
1 Foreman.
2 Deposit melters.
1 Gold ingot melter.
1 Silver melter.
1 Fine gold boiler.
1 Silver reducer.
1 In charge of corroding houses.
1 ,, ,, hydraulic press.
1 ,, ,, precipitating tanks.
1 Granulation melter.
2 Sweep cellar operatives.
1 Messenger.
14 Helpers.

Coiner's Department.

2 Clerks.
1 Die hardener.
6 Foremen of work rooms.
1 Pressman.
2 Weighers.
2 Cutters.
1 Whitener.
1 Machine hand.
11 Workmen.
28 Adjusters.
1 Janitress.

Mint at Carson.

Superintendent.
Assayer.
Melter and refiner.
4 Watchmen.

Mint at New Orleans.

Superintendent.
Assayer.
Melter and refiner.
Coiner.
Chief clerk.
Cashier.
Assistant assayer.
,, melter and refiner.
,, coiner.
Bookkeeper.
Weigh clerk.
Abstract ,,
Assayer's computing clerk.
Warrant clerk.
Assistant weigh clerk.
Cashier's clerk.

General Department.

2 Clerks.
1 Copyist.
1 Assistant bookkeeper.
1 Adjuster of scales.
1 Deposit melter.
1 Storekeeper.
1 Plumber.
1 Engineer.
1 Machinist.
1 Carpenter.

MINT AT NEW ORLEANS—*continued.*

1 Blacksmith.
1 Blacksmith's helper.
2 Conductors.
1 Doorkeeper.
1 Fireman.
1 Yardman.
1 Elevator hand.
5 Watchmen.
10 Workmen.
1 Scrubwoman.

Assayer's Department.

Foreman.
Weigher.

Melter and Refiner's Department.

1 Clerk.
1 General foreman.
1 Foreman, melting room.
16 Workmen.

Coiner's Department.

1 Foreman, coining room.
1 Pressman.
1 Counter.
2 Weighers.
1 In charge of mills.
6 Rollers.
2 Annealers.
4 Cutters.
4 Cleaners.
15 Workmen.
45 Adjusters.

ASSAY OFFICE AT NEW YORK.

Superintendent.
Assayer.
Melter and refiner.
Chief clerk.
Assistant melter and refiner.
Weigh clerk.
Bookkeeper.
Warrant Clerk.
First assistant assayer.
Second ,, ,,
Third ,, ,,
Cashier.
Bar clerk.
Abstract clerk.
Assayer's computing clerk.
Assistant weigh clerk.
Register of deposits.

General Department.

3 Clerks.
1 Janitor.
1 Messenger.
1 Watchman.
3 Workmen.

Assayer's Department.

1 Chemist.
1 Foreman.
3 Weighers.
1 Assayer's sampler.
1 Cupel maker.
1 Workman.

Melter and Refiner's Department.

1 Foreman.
1 Pressman.
1 Mason.
1 Plumber.
1 Engineer.
1 In sweep, cellar.
5 Watchmen.
7 Helpers.
10 Melters.
4 In acid room.
1 ,, reducing room.

MINT AT DENVER.

Assayer in charge.
Melter.
Chief clerk.
Weigh ,,
Clerk.
Assistant assayer.
3 Assistants, assayer's department.
2 ,, melting ,,
1 Assistant calculation clerk.
1 Day watchman.
1 Night watchman.
1 Labourer.

ASSAY OFFICE AT HELENA, MONT.

Assayer in charge.
Melter.
Chief clerk.
Clerk.
Cashier.
2 Assistants, assayer's department.
1 Foreman, laboratory.
1 Assistant, melting department.
1 Deposit clerk.
2 Watchmen.

ASSAY OFFICE AT BOISE CITY, IDAHO.

Assayer and melter.	Assistant assayer.
Clerk.	Watchman.
Assistant melter.	

ASSAY OFFICE AT CHARLOTTE, N.C.

Assayer and melter.	Assistant assayer.

ASSAY OFFICE AT ST. LOUIS, MO.

Assayer in charge.	Workman.
Clerk.	Janitor.

PARTING GOLD AND SILVER IN CALIFORNIA.

THE GUTZKOW METHOD.

In the year 1867 Mr. F. Gutzkow invented a process for parting gold and silver, which he introduced into the San Francisco Assaying and Refining Works, which is not only original but of great economic value, as being at the same time one of the most expeditious and one of the least expensive methods of parting in use.

The works in San Francisco were situated in the outskirts of the city, near the lines of railway, and within a short distance of the water, and were adapted to treat about one ton per day of the gold and silver bullion of California, Nevada, and the adjacent territories, as well as that sent from Mexico, China, and Japan. They frequently, also, parted for the United States Mint, in San Francisco, when there was a press of work there.

The material comes as bricks and bars, stamped with their assay value, and as coin. This material may be silver brick containing only a little gold, as that from the Reese River district, or a large quantity of it, as that from the Comstock lode; or it may be the gold of California, with but little silver or other impurity. This gives three kinds of material to be subjected to parting:

1st. The gold bars of California.

2nd. The silver bricks from the Comstock, rich in gold.

3rd. Material containing a considerable percentage of copper.

The first are inquartated so as to contain two parts of gold to three of silver, and are granulated. The second are dissolved as bricks, just as they come from the mine, without granulation; they generally contain from 2 to 10 per cent. of gold. The third is

usually the bricks which come from the tail mills and some of the mines of Nevada, and include certain classes of coins. These are melted with high-grade bars until the proportion of copper is not over 8 to 12 per cent. They are not, however, granulated. It will thus be seen that the methods are from the outset quite different from those in use elsewhere, where granulation of all the alloys is considered indispensable, and forms the first and the essential step in all the different varieties of processes.

The melting is done in graphite crucibles in an anthracite furnace with a forced draught. Each charge is prepared and inquartated separately.

The metal is poured by hand from a ladle into cold water, in a very small stream, a circular motion in different planes being given to it while pouring, it being kept from .06 to .09 metres above the water. The water is contained in a large tub, and has a rapid rotary motion given to it. The metal assumes an irregular shape, more or less spherical, called granules. These are carefully dried and are then ready for solution.

The process, which is the simplest of all the parting processes, consists of eight operations:

1. The solution of the alloy.
2. Settling the solution.
3. Crystallisation of the silver sulphate.
4. Decomposition of the silver sulphate by iron sulphate.
5. Purifying, pressing, and melting the silver.
6. Treatment of the pot residues.
7. „ „ residues of the settling tanks.
8. „ „ sesquioxide of iron sulphate.

I. Solution of the Alloy.

The pots R, Fig. 119, in which this solution is effected are always made of cast iron. The platinum basins, which were formerly used exclusively in European works, are not used here, Cast iron will resist the action of a very concentrated sulphuric acid for a long time, but as the acid is at times diluted, pots made of ordinary cast iron will not last very long. In the Vienna Mint the iron used for making the pots contains from 3 to 4 per cent. of phosphorus. The iron is very white and compact, and the

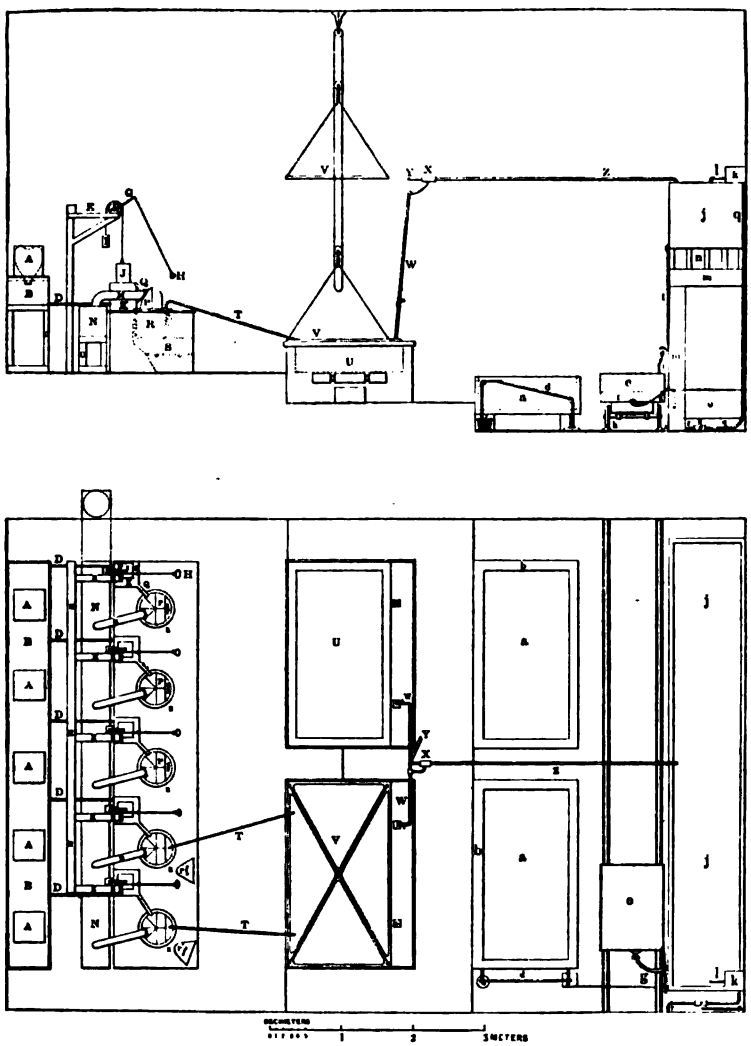


FIG. 119. THE GUTZKOW PARTING WORKS AT SAN FRANCISCO, CALIFORNIA.

To face page 776.

resistance to the action of the acid is probably due to its very close texture. Even with only 2 per cent. of phosphorus these pots have been known to last for fully two years. Silicon in large quantities produces the same effect. The late Dr. J. Lawrence Smith cites a ferro-silicon with 16 per cent. of silicon, which was not appreciably attacked by *aqua regia*, which, if it could be cast to requisite shape, might be used indefinitely for the solution of gold and platinum. The pots in San Francisco are .66 metre in diameter, .45 metre deep, and .02 thick. They are made with flat bottoms, and are set in brickwork, five in a row, each one having its own fireplace.

It is impossible to have the acid always of exactly the same strength, and it also varies in strength at different times in the operation, so that the pots are slightly acted upon and lose usually from 60 kilos. to 70 kilos. a year in weight. They wear down in from twenty to twenty-four months to from .05 to .06 metre in thickness.

The usual charge of bullion is from 100 kilos. to 150 kilos., depending on whether it is made of granulated material or of bricks. The larger charge is for bricks. The works contain two rows or benches of these pots R, each one of which has its own cover P, and all are covered with a hood, which is open over the whole length of the bench, but is set low enough to prevent any escape of acid fumes into the works. It would be more economical if the pots were placed six in a row, as they could be taken care of by the same number of men as now take care of five, and if they were somewhat larger, for then more alloy could be treated by the same number of men in less time. In the conical hood over each pot there is an opening on the front for introducing the charge, which is protected by a cover P^1, and another on the back, from which a pipe leads to a lead-lined conduit M behind the furnace for condensing the escaping vapours of sulphuric acid. The cover of the opening for introducing the charge slides on the outside and is moved by a wrought-iron handle cast into it. Each bench of solution pots is provided with its own condensing chamber N, which is 4.6 metres by 4.6 metres, and 11.6 metres long. It is made of sheet lead supported by a framework of wood. The sulphuric acid which

escapes from the condensation chamber is carried into a tower, and from that it is discharged into a high chimney. The sulphuric acid is introduced into the pot as needed, and is at a density of 66 deg. B. Its depth in the pot is from 0.23 to 0.25 metre. The supply of acid comes from a tank B, outside of the building, and is regulated by a lever H, placed immediately over the pot. Each pot has its own supply reservoir. The carboys of acid A are placed in the tank B upside down outside the building, and are left so until they empty themselves. They dip into the acid in the tank about 0.15 metre. The acid tank connects with a lead-lined reservoir K placed between and behind the pots by a pipe D, so that the acid in both is at the same level. Into this reservoir a plunger J of wood covered with lead is fitted with a play of .03 to .04 metre on every side, and is counterpoised by a weight I. The plunger is raised and lowered by means of an iron handle on the side of the pot, which is within easy reach of the workman. When the handle is pulled down the plunger descends into the reservoir and forces an amount of acid corresponding to the volume of the plunger into the pot. When the handle is released the counterpoise raises the plunger and the reservoir is again filled with acid, that amount only which has passed into the pots escaping from the carboys in the tank. The size of the plunger is so regulated that about 10 kilos. of acid are discharged each time the plunger is entirely lowered. More or less can be discharged at any time by regulating the descent of the handle. The pot may thus be filled with acid to any required depth in a very few minutes, the acid being forced over into the pot each time the handle descends. The supply of acid is introduced generally about once in every fifteen minutes when the metal is first charged. The first charge of acid is introduced by the fireman, who comes an hour before the refiners and lights the fire, so that by the time they arrive the acid is boiling. The refiner removes the cover from the pot, and when granules are used introduces three boxes containing from 30 kilos. to 40 kilos. of the granulated metal or a little larger weight of bars or bricks. When all the charge is made the cover is replaced and the charge left to itself from three to four hours, depending upon the size of the granules or bricks.

The acid is kept boiling during the whole operation, but care is taken not to allow it to boil so violently that any projections or spattering will take place. It is necessary to remove the cover from time to time to stir a charge of granulated metal and prevent it from packing. One refiner and one fireman do all the work connected with the pots.

II. Settling the Solution.

The solution in the pots is heated to prevent its crystallising. When ready it is syphoned into an iron tank U, Fig. 119, 2.75 metres long by 1.5 metres wide and 0.5 metre deep, containing sulphuric acid at 58 deg. B., and at a temperature of 110 deg. Cent., 0.5 cubic metre being required for every 100 kilos. of metal treated. The acid used is the mother liquor resulting from the drainage of the crystals of silver sulphate. The syphoning is done by means of a vacuum formed by steam. The iron tanks are covered with a heavy iron lid V, which when necessary is raised by a Weston pulley. To make the lid air-tight, it rests on a broad india-rubber band. One end of an iron gas-pipe T, bent to the proper form, is placed in the pot, and the other, fitted air-tight by means of the rubber packing, passes through the cover of the vat. Each bench of pots has its own vats. All the pots discharge in the same way. On the opposite side a steam-pipe connects in such a way that a vacuum is produced in the tank, and transfers the acid from the pots to the tank. This method is much preferable to the old system of ladling, and the work is done in a much shorter time. The steam used to produce the vacuum passes into the tank *j* for holding the iron sulphate solution to heat it, so that none of it is lost. Previous to syphoning, the gold at the bottom of the pots is pushed back to one side, and the pipe is introduced near the bottom, away from the gold residue, so as to empty them as dry as possible. The tank into which the liquor discharges is filled to within .05 metre to .08 metre of the top. The acid from the pots, with the silver in solution, is diluted so as to bring it from 58 deg. B. to 66 deg., and is again covered without the use of the india-rubber band and left to settle.

The liquor contains silver sulphate, iron sulphate, and

graphite from the solution pots, copper sulphate, and, if there was any lead in the metal, some lead sulphate, most of which, with any very fine gold carried over in the baling, will be in suspension. The liquor is quite turbid from all these substances, and the object of adding the water is to clarify it as the lead sulphate becomes less soluble, and its precipitation helps to clear the liquor from any particles, especially of gold in suspension. This tank is placed over a fireplace heated with wood, and is kept heated during the time that the liquor is being clarified, and subsequently discharged from this tank.

III. Crystallisation of the Silver Sulphate.

When the liquor is entirely clear it is syphoned into another tank of iron *a* of the same size, but twice as deep. This tank is placed inside of a leaden one. Water flows between the two to cool down the liquor as rapidly as possible. The lead tank serves also the purpose of catching any of the silver sulphate in case of a leak in the iron tank. The solution is clarified and siphoned into the tank about 10 a.m. By 7 p.m. the saturated solution of silver sulphate is thoroughly cool and has crystallised. The supernatant mother liquor, now reduced to a temperature between 30 deg. and 40 deg. Cent., contains all the copper sulphate. It is forced back into the first vat U from which the precipitate has not been removed. This is done by making the cover of the first vat hermetically tight, and producing, as before, a partial vacuum with steam, on the Giffard injector principle. The acid is thus removed without difficulty and is used again as acid at 58 deg. B., which is heated in the vat to receive the pot liquor.

In order to effect the separation of the mother liquor so that the crystals of silver sulphate may be as free from acid and as dry as possible, the bottom of the crystallisation tank has a well *c* into which the mother liquor drains, and which also serves to bring the syphon to the lowest point. After all the acid has been removed by the syphon it is used as a drip, by means of a stop-cock placed in the bottom, which opens into a vessel placed beneath the vat. It is desirable, in order to facilitate the reduc-

tion of the crystals of silver sulphate, that they should be as free from acid as possible.

The silver sulphate is found in the bottom of the tank crystallised to a depth of about .05 metre. It is a yellow incrustation, which is more or less hard, and scattered through it there is a reddish powder of copper oxide. It is very nearly free from acid, and if in the bottom of the tank there is any mother liquor left it must be carefully drained.

IV. Decomposition of the Silver Sulphate by Iron Sulphate.

The crystals are now removed from the vat with an iron shovel and placed in a wooden box *e* lined with lead, 1.20 by .90 metres and .30 metres deep. One such box is required for every five solution pots. Between the false and the real bottom a stopcock is placed. This vat is on a truck, and can be easily moved from one part of the works to another. After the crystals have been carefully drained a saturated solution of protoxide of iron sulphate in water heated by steam is introduced. This solution should be made as nearly neutral as possible. It flows upon the mass from a tank *j* at a higher level and runs through the crystals, draining into the false bottom, from which it flows into a large tank 5.6 metres by 2.8 metres and .90 metres deep. The larger the tank the better. The first material dissolved is the copper, the solution of which is kept in a separate tank. This liquor contains some silver, which is recovered from it, and is blue, owing to the presence of copper. As soon as it becomes brown it is run into the regenerating tank. The green solution of protoxide of iron sulphate dissolves out first the salts of copper, attacks the silver salt, reducing it to a metallic state, and, taking up the acid set free, becomes sesquioxide of iron sulphate, which is brown. This operation is continued until the liquid remains permanently green, which shows that no action is taking place and that the silver is all reduced to a metallic state. The only precaution required during the operation is to to turn the crystals of the silver salt over from time to time in order to expose fresh surfaces to the iron sulphate. While the liquor is running off and is quite brown nothing need be done. When it becomes

slightly greenish the crystals must be stirred. When it is persistently green the operation is finished. This operation takes one man from four to five hours; 11.5 litres of the saturated solution of green vitriol are required for every kilogramme of silver sulphate treated.

The solution of iron sulphate in passing over the crystals of silver sulphate dissolves some of it, amounting generally to 7½ per cent., which is regained on cooling. Both the blue and the brown liquors are caught in separate lead-lined vats, and the silver in the blue liquor separated by copper, and that in the brown precipitated by iron. The amount of silver contained in the brown solution amounts to 2½ per cent. of the original charge. All the copper is afterwards regained from the blue solution. The silver which is reduced remains in the box as a more or less coherent mass of metallic silver, which has kept the shapes of the crystals of the sulphate. It amounts to 90 per cent. of the original silver charge. The reduction is very rapidly effected, the contents of the five pots being reduced in from three to four hours.

V. Purifying, Pressing, and Melting the Silver.

The silver removed from the box is liable to contain some silver sulphate. It is not desirable, on account of the quantity of liquor required to reduce it all, to continue the operation until every trace of sulphate has been reduced. It is charged into another lead-lined vat with a false bottom, which is on wheels, and is 1.22 by .90 metres and .90 metres deep. It is intended to hold the silver from both of the decomposition vats. It is spread in layers of .025 metre thick, and over each layer a sheet of copper is laid, and so the vat is charged nearly to the top. It is then filled in the evening with hot water and left until the following morning. Any silver sulphate which has not been acted upon by the protoxide of iron sulphate is dissolved out and decomposed by the copper plates. In the morning the liquor is drained off and the silver transferred to a filter lined with drilling. Hot water is run through it and tested for silver. If any is found the silver in the filter is washed until every trace of sulphate is removed and the filtrate put one side for the

subsequent treatment. If not, the mass in the filter is washed as long as it shows any trace of copper, iron, or acid. This takes one to two hours. When the silver is quite free from any impurities it is said to be *sweet*. It is then carefully dried and pressed in a hydraulic press into cakes 0.30 metre in diameter and 0.25 in thickness, which weigh from 15 to 20 kilogrammes These cakes are dried at a high heat and then melted in graphite crucibles into bricks weighing about 40 kilogrammes. They are 998 fine. The silver could also be made 999 or 1000 fine, but this is not desirable, since no allowance is made at the mint for such fine silver, because it is not wanted for commercial purposes finer than 998. It is therefore never made finer. It is generally made of whatever fineness is called for.

VI. Treatment of the Pot Residues.

After the silver sulphate has been all syphoned out of the solution pots, gold remains behind as a residue in a more or less finely divided state. When granulated metal has been used it is boiled at once from one and a half to two hours with fresh acid, which is then put to one side to be used on a fresh charge of metal. When silver bricks have been treated, the gold is already 996 fine, and no second boiling is required. The gold is dipped out with a cast-iron strainer into a cast-iron pot, on wheels, containing hot acid in sufficient quantity to cover it. The residue which cannot be reached with the ladle is taken out with a hoe-shaped tool. A very small quantity is still left in the pot, which is, however, no loss, since it is recovered in the next operation. The gold is then transferred into a tank on wheels of about the same size as the tanks for washing the silver, but with a vertical partition. The compartment for holding the gold has a false bottom, and comprises about three-fourths of the tank. Acid is baled upon the gold, which goes through the false bottom, and is baled out on the other side. This acid is distributed in the solution pots the next day. About half a potful of acid is run through in about ten minutes. The gold is then put upon a wooden filter and hot distilled water is run through it until it is entirely sweet. As this water dissolves a very small amount of silver, it is always saved. The gold, when

sweet, goes to the press, and is dried and melted into ingots like the silver. It is made 900 to 993 or even 994 fine.

VII. Treatment of the Residues of the Settling Tanks.

The residue in the tanks in which the silver is settled consists of the lead and iron sulphates, the graphite separated from the solution pots, some very fine gold which has been carried over in the syphoning, and some silver sulphate. It is allowed to accumulate for a month, at which time it will be about 0.05 metre in thickness on the bottom of the tank. When it is to be collected the mother liquor from the silver sulphate solution is pumped up as before, heated, and drawn off as usual, but no solution from the pots is added. The residue at the bottom of the tank is then carefully washed with hot water to dissolve out any silver sulphate, and treated with granulated zinc to reduce the lead sulphate. It must be treated to separate the precious metals. It is then collected, dried, and fused with carbonate of soda, to remove the graphite, in a reverberatory furnace, with the slags removed from the fusion of the silver and gold, and is then cupelled. The litharge from the cupellation is not reduced; the very small quantity produced is sold.

The ashes from the fireplace of the bullion furnace are ground and washed, and what metal can be taken out in this way is separated. All the heavy material is treated in the reverberatory furnace at the same time.

VIII. Treatment of the Sesquioxide of Iron Sulphate.

The brown liquor which has passed over the silver sulphate and been caught in a tank when sufficiently cool, is transferred to a lead-lined vat of about the same size as that previously used, and brought to the condition of protoxide of iron sulphate, with scrap iron. The iron precipitates both copper and silver, which fall to the bottom and form a muddy mass. The solution of sulphate, as soon as it is clear, is forced up again to the tank to be used on the silver sulphate, and this regeneration goes on indefinitely. The mud in the bottom of the tank is collected every week and put into the silver sulphate vat, and the protoxide of iron sulphate run through it.

The copper is first dissolved as sulphate, and so long as the liquor is blue it is kept separate, and is added to the first liquor obtained in acting on the silver sulphate. The silver is precipitated from it with copper. The liquor which is not blue is run back to the regenerating tank with the rest of the iron liquor. As the volume of the solution of the protoxide of iron sulphate is being constantly increased by fresh additions of water, iron, and acid, it accumulates beyond the amount required for the process. Once a week the excess of this liquor, freed from both copper and silver, is allowed to run into the sewer The bluish liquor, rich in copper after the silver has been precipitated from it by copper, is run into another tank, and the copper precipitated with iron. The cement copper is dried and oxidized in a reverberatory furnace, dissolved in dilute sulphuric acid, crystallized, and sold. The quantity of this salt produced is very small, as it only corresponds to the copper in the bullion and the very small amount dissolved in the use of sheet-copper used to precipitate the silver.

The cement silver is collected, washed, dried, and treated as before. All the transfers of liquor are done on the injector principle. All of them except the acid are kept hot by steam; the acids are heated by fires. The number of men required to do the work are two melters, two melters' helpers, two refiners, two refiners' helpers, two reducers, one sweetener and drier, one pressman, twelve men working eight hours a day to refine one ton of bullion, whether gold or silver. Some of these men are at times doing other work. The works could treat about $250,000 of gold and silver together, or about 500 kilogrammes of gold in a day.

These three processes are the only parting processes in use in the United States to my knowledge, the very simple precipitation by iron in use at Lautenthall* not having been introduced in the United States.

The following pages give the production and consumption of gold and silver in the world, and are supplementary to those

* School of Mines Quarterly, vol. vi., p. 240.

given in the introduction of vol. i. The information has been taken for the most part from the various reports of the Director of the United States Mint.

The following Table* gives the bullion production of the mines of California for the twelve months ending January 1, 1886.

County.	Gold.		Silver.		Total.
	Reported.	Unreported.	Reported.	Unreported.	
	$	$	$	$	$
Amador	1,835,591.23	310,000.00	406.40	...	2,145,997.63
Butte	422,568.50	250,000.00	...	3,700.00	676,268.50
Calaveras	397,538.23	130,000.00	2,558.20	...	530,096.43
Colusa	40,000.00	...	5,000.00	...	45,000.00
Del Norte	24,390.00	15,000.00	9.18	...	39,399.18
El Dorado	383,353.85	35,000.00	...	...	418,353.85
Fresno	67,500.00	7,000.00	2,456.00	...	76,956.00
Humboldt	20,130.37	9,600.00	...	...	29,730.37
Inyo	7,498.04	17,500.00	59,961.49	13,500.00	98,459.53
Kern	22,003.00	50,000.00	...	...	72,003.00
Lassen	7,500.00	7,500.00	150.00	...	15,150.00
Los Angeles	...	22,500.00	445.00	1,500.00	24,445.00
Mariposa	87,177.00	62,000.00	100.00	...	149,277.00
Merced	...	10,000.00	...	...	10,000.00
Modoc	...	60,000.00	...	...	60,000.00
Mono	477,860.41	5,000.00	88,349.49	3,500.00	574,709.11
Napa	...	...	5,000.00	...	5,000.00
Nevada	1,138,873.68	1,439,000.00	4,835.43	...	2,582,709.90
Placer	507,801.51	398,500.00	411.43	...	906,712.94
Plumas	688,307.71	152,000.00	...	...	840,307.71
Sacramento	253,522.00	100,000.00	...	...	353,522.00
SanBernardino	3,000.00	20,000.00	2,283,436.37	80,000.00	2,386,436.37
San Diego	71,125.00	20,000.00	...	2,000.00	93,125.00
San Joaquin	...	2,500.00	...	...	2.500.00
Shasta	236,004.75	181,000.00	9,123.38	100,000.00	426,228.13
Sierra	1,060,380.97	373,500.00	10.54	...	1,433,891.51
Siskiyou	119,658.70	219,000.00	...	...	338,658.70
Stanislaus	18,660.00	...	...	...	18,660.00
Trinity	175,347.69	162,800.00	10.00	...	338,157.69
Tulare	...	7,500.00	...	...	7,500.00
Tuolumne	149,503.34	171,400.00	1,273.00	200.00	322,376.34
Yuba	132,448.73	75,000.00	...	...	207,448.73
Total	8,347,744.71	4,313.300	2,463,535.91	104,500.00	15,229,080.62

The following Table† gives the distribution, in round numbers, by States and territories, of the estimated total production of precious metals in the United States for 1885.

* Prod. of the Precious Metals in the U.S. for 1885, p. 131.

† Report of the Director of the Mint for 1886, p. 51.

State or Territory.	Gold.	Silver.	Total.
	$	$	$
Alaska	300,000	2,000	302,000
Arizona	880,000	3,800,000	4,680,000
California	12,700,000	2,500,000	15,200,000
Colorado	4,200,000	15,800,000	20,000,000
Dakota	3,200,000	100,000	3,300,000
Georgia	136,000	...	136,000
Idaho	1,800,000	3,500,000	5,300,000
Montana	3,300,000	10,060,000	13,360,000
Nevada	3,100,000	6,000,000	9,100,000
New Mexico	800,000	3,000,000	3,800,000
North Carolina	152,000	3,000	155,000
Oregon	800,000	10,000	810,000
South Carolina	43,000	...	43,000
Utah	180,000	6,750,000	6,930,000
Washington	120,000	70,000	190,000
Texas, Alabama, Tennessee, Virginia, Vermont, Michigan, and Wyoming	90,000	5,000	95,000
Total	31,801,000	51,600,000	83,401,000

The following Table gives a comparison of estimates of production of the United States, by States and territories, for 1884 and 1885.

State or Territory.	Gold.		Silver.		Total.	
	1884.	1885.	1884.	1885.	1884.	1885.
	$	$	$	$	$	$
Alaska	200,000	300,000	...	2,000	200,000	302,000
Arizona	930,000	880,000	4,500,000	3,800,000	5,430,000	4,680,000
California	13,600,000	12,700,000	3,000,000	2,500,000	16,600,000	15,000,000
Colorado	4,250,000	4,200,000	16,000,000	15,800,000	20,250,000	20,200,000
Dakota	3,300,000	3,200,000	150,000	100,000	3,450,000	3,300,000
Georgia	137,000	136,000	...	...	137,000	136,000
Idaho	1,250,100	1,800,000	2,720,000	3,500,000	3,970,000	5,300,000
Montana	2,170,000	3,300,000	7,000,000	10,060,000	9,170,000	13,360,000
Nevada	3,500,000	3,100,000	5,600,000	6,000,000	9,100,000	9,100,000
New Mexico	300,000	800,000	3,000,000	3,000,000	3,300,000	3,800,000
North Carolina	157,000	152,000	3,500	3.000	160,500	155,000
Oregon	660,000	800,000	20,000	10,000	680,000	810,000
South Carolina	57,000	43,000	500	...	57,500	43,000
Utah	120,000	180,000	6,800,000	6,750,000	6,920,000	6,930,000
Washington	85,000	120,000	1,000	70,000	86,000	190,000
Texas, Alabama, Tennessee, Virginia, Vermont, Michigan and Wyoming	84,000	90,000	5,000	5,000	89,000	95,000
Total	30,800,000	31,801,000	48.800,000	51,600,000	79,600.000	83,401,000

Colorado still retains the foremost rank as the largest producer of the precious metals in the United States. Notwithstanding the decline in the price of silver, the quantity produced remained about the same as during the preceding year. California with its $12,700,000 gold and $2,500,000 silver retains second position. The same notable changes are seen in the production of Montana and Idaho, the former having increased from $9,170,000 in 1884 to $13,360,000 in 1885, and the latter from $3,970,000 in 1884 to $5,300,000 in 1885. Nevada, Utah, New Mexico, and Dakota still hold their own, while the production of Arizona has slightly decreased.

The following Table* gives the annual† average production of the precious metals in the world since the discovery of America.

Periods.	Silver.	Gold.	Value of Silver and Gold.
	kilogrammes.	kilogrammes.	million francs.
1493-1520	47,000	5,800	30.7
1521-1544	90,200	7,160	45.2
1545-1560	311,600	8,510	99.7
1561-1580	299,500	6,840	91.2
1581-1600	418,900	7,380	120.0
1601-1620	422,000	8,520	124.9
1621-1640	393,600	8,300	117.5
1641-1660	366,300	8,770	113.0
1661-1680	337,000	9,260	108.1
1681-1700	341,900	10,765	114.5
1701-1720	355,600	12,820	124.7
1721-1740	431,200	19,080	163.5
1741-1760	533,145	24,610	205.7
1761-1780	652,740	20,705	219.1
1781-1800	879,060	17,790	259.9
1801-1810	894,150	17,778	263.2
1811-1820	540,770	11,445	161.6
1821-1830	460,560	14,216	153.2
1831-1840	596,450	20,289	205.0
1841-1850	780,415	54,759	366.6
1851-1855	886,115	197,515	882.2
1856-1860	904,990	206,058	922.2
1861-1865	1,101,150	198,207	893.4
1866-1870	1,339,085	191,900	970.5
1871-1875	1,969,425	170,675	1,038.3
1876-1880	2,470,440	171,020	1,152.9
1881-1882	2,578,000	158,250	1,131.9
1883	2,895,520	143,940	1,139.2
1884	2,860,000	140,000	1,117.7

* Report of the Director of the U.S. Mine for 1886, p. 171.

† Dr. Adolf Soetbeer.

The total production from 1493 to 1875 amounted to 9,453,345 kilogrammes of gold and 180,511,485 kilogrammes of silver, giving, on the basis of the ratio of 15½, a value of 73 milliards 584 million of francs. With the production of the seven succeeding years we have, for the period 1793-1884, the following figures:

Total weight of gold, 10,908,885 kilogrammes.
,, ,, silver, 203,951,205 kilogrammes.
,, value of gold and silver, 83,865 millions of francs.

The various estimates of the production of silver in the United States from 1871 to 1876, inclusive, adduced by Mr. Del Mar on the part of several authorities, comprising, I believe, all of note at the date of his report, February 24, 1877, are as follows:*

COMPARATIVE TABLE SHOWING SEVERAL PUBLISHED ESTIMATES OF THE PRODUCTION OF SILVER IN THE UNITED STATES DURING THE CALENDAR YEARS 1871–1876, INCLUSIVE.†

Authority.	1871.	1872.	1873.	1874.	1875.	1876.
[1] San Francisco Journal of Commerce	dols. 24,246,000	dols. 27,548,811	dols. 38,500,000	dols. 40,250,000	dols. 46,500,000	dols. 48,000,000
[2] Whitehill, mineralogist of Nevada	23,000,000	23,000,000	35,000,000	40,000,000	40,000,000	
[3] San Francisco Commercial Herald	25,000,000	28,000,000	36,000,000	37,000,000	38,000,000	45,000,000
[4] Raymond, United States Commissioner of Mines	22,000,000	25,750,000	36,500,000	32,800,000	41,400,000	
[5] Gansl's report to N. M. Rothschild and Sons	20,250,000	25,000,000	32,500,000	30,000,000	35,000,000	50,000,000
[6] Valentine's amended circular of December 30, 1876	20,286,000	20,527,500	28,352,100	30,498,000	34,043,910	41,506,672
[7] Valentine's amended estimates of February 5, 1877	19,286,000	19,924,429	27,483,302	29,699,122	31,635,239	39,292,924
[8] Returns of the mines to the United States Monetary Commission, 1876, 1877	18,111,351	18,556,577	25,131,460	25,402,382	30,209,986	38,184,350

[1] According to the San Francisco Journal of Commerce, January 12, 1876, and January 24, 1877, from time to time amended.
[2] According to Mr. Whitehill, in his report of State mineralogist of Nevada, Carson City, 1875.
[3] According to the San Francisco Commercial Herald of January 18, 1877.
[4] According to Mr. Raymond, as published in Hon. Abram S. Hewitt's address on mining and metallurgy, before the American Institute of Mining Engineers, Philadelphia, June 20, 1876.
[5] According to Mr. Gansl's report to Messrs. N. M. Rothschild and Sons, London, as published in the appendix to the British Silver Commission report.
[6] According to Mr. Valentine's amended circular of December 30, 1876.
[7] According to Mr. Valentine's amended estimates to the chairman of the United States Monetary Commission, dated February 5, 1877.
[8] According to returns made by the mining companies, &c., for the United States Monetary Commission, between November, 1876, and February, 1877, the returns for 1876, completed to the close of the year.

The production of gold in the world has remained almost constant for a number of years, as will be seen from the following Table:

GOLD.—PRODUCTION OF THE WORLD.

Calendar Years.	Amount.	Years.	Amount.
	$		$
1880 . . .	106,000,000	1883 . . .	97,000,000
1881 . . .	103,000,000	1884 . . .	99,000,000
1882 . . .	102,000,000	1885 . . .	101,500,000

* Production of Precious Metals in the U.S., 1885, p. 46.

† For a continuation of this Table see vol. i., pp. 48 and 50.

It is a curious fact that, notwithstanding the large depreciation in its value the production of silver in the world has progressively increased, as shown by the following Table:

SILVER.—PRODUCTION OF THE WORLD.

Years.	Amount.	Years.	Amount.
	$		$
1880 . . .	96,700,000	1883 . . .	117,000,000
1881 . . .	102,000,000	1884 . . .	116,500,000
1882 . . .	111,000,000	1885 . . .	124,900,000

The production of gold in 1884 was stated for the fiscal year 1885 to have been, in round numbers, $95,000,000. But official returns show that the production of Australasia was over $2,000,000 more than the amount estimated, based upon deposits at the Melbourne and Sydney mints; and of Venezuela $1,300,000 more; thus increasing the aggregate of production for 1884 to almost $99,000,000. The production of gold for 1885 approximated $101,500,000, which is an increase of $2,500,000 from 1884.

The increase in the production of silver from $116,500,000 in 1884 to nearly $125,000,000 in 1885 was principally on the part of the United States and Mexico, the production of the former being $2,800,000 more than in 1884, and of the latter about $5,000,000.

Australia, for the first time, also appears as a producer of a considerable quantity of silver. Heretofore Australia has been credited only with the silver parted from gold deposits at the Melbourne and Sydney mints. The official statistics for 1885 show a production of silver valued at $1,048,279.

The United States still preserves the first rank among the nations of the world as the largest producer of the precious metals, its production of gold and silver for the calendar year 1885 having reached the sum of $83,400,000, out of a total production of $219,000,000, or about two-fifths of the production of the world. The neighbouring republic of Mexico occupies second place, with a recorded production of over $32,000,000. Australia comes next, with a production of nearly $31,000,000. Russia takes fourth rank, with a production of $26,000,000, principally gold.

Bolivia is also a large producer, the production of silver in that republic being officially returned at $16,000,000.

The following Table* gives the world's production of gold and silver for 1885.

Countries.	Gold.		Silver.	
	kilos.	$	kilos.	$
United States	47,848	31,800,000	1,241,578	51,600,000
Russia	38,125	25,338,218	15,554	646,424
Australasia	44,877	29,824,949	25,225	1,048,279
Mexico	1,304	866,671	772,661	32,111,778
Germany	*k*611	406,071	*k*230,339	9,572,888
Austria-Hungary	*h*1,658	1,101,907	*h*49,424	2,054,061
Sweden	*h*19	12,627	*h*1,816	75,472
Norway	...	...	*h*6,387	265,490
Italy	*d*109	72,375	*d*432	17,949
Spain	...	...	*h*3,562	148,003
Turkey	*10	6,646	*2,164	89,916
Argentine Republic	*f*118	78,546	*f*10,109	420,225
Colombia	*5,802	3,856,000	*18,286	760,000
Bolivia	*g*109	72,375	‖384,985	16,000,000
Chili	*h*500	332,300	*h*160,000	6,649,600
Brazil	898	596,740		
Japan	17	11,006	23,089	959,560
Africa	*m*2,500	1,661,500		
Venezuela	7,033	4,674,132		
Dominion of Canada	*n*1,084	720,650		
France	...	...	*h*5,905	245,412
Peru	226	150,000	47,822	1,987,500
Great Britain	...	35	7,608	316,230
Total	152,848	101,562,748	3,006,946	124,968,784

* Estimated same as officially communicated for 1882.
‖ ,, ,, ,, 1883.
d ,, ,, ,, 1877.
f ,, ,, ,, 1879.
h ,, ,, ,, 1884.

k Officially communicated. Includes foreign ore smelted. Production of Prussian States officially reported in "Zeitschrift f. d. Berg-Hütten-und-Salinen Wesen" as: Gold, kilos., 130; silver, 195,034.

m Estimated from imports of gold to England and India.

n Financial and Mining Record, July 17, 1886.

The following Table exhibits the annual estimated consumption of gold and silver in the arts by the principal nations of the world. It does not include any consumption by India or the Eastern countries generally, or by Mexico or South American countries.†

* Report of the Director of the U.S. Mint for 1886, p. 169.

† *Ibid.*, p. 61.

* ANNUAL INDUSTRIAL CONSUMPTION OF GOLD AND SILVER BY THE PRINCIPAL NATIONS OF THE WORLD, FROM LATEST AUTHORITY.

Countries.	Population.	Gold (fine ounces).	Value.	Silver (fine ounces).	Coining value. ($1.2929)	Total value gold and silver.	Per capita.	
							Gold.	Silver.
			$		$	$	$	$
United States (Burchard)	58,000,000	626,925	13,000,000	3,697,250	4,000,000	17,000,000	0.22 4/10	0.07
England (mean of several authorities)	36,000,000	546,550	11,500,000	2,604,150	3,000,000	14,500,000	.32	.08 3/10
France (Dumas)	37,000,000	401,875	8,600,000	2,411,250	2,800,000	11,400,000	.23 8/10	.07 8/10
Germany (Soetbeer)	45,000,000	385,800	8,200,000	2,636,300	3,000,000	11,200,000	.18 2/10	.06 6/10
Switzerland (Lardy) †	2,846,000	321,500	6,600,000	835,900	1,000,000	7,600,000	2.31 8/10	.35
Austria-Hungary (Nibauer)	37,800,000	...	...	835,900	1,000,000	1,000,000	...	.02 9/10
Other countries (Soetbeer)	230,000,000	450,100	9,500,000	3,697,250	4,000,000	13,500,000	.04 1/10	.01 7/10
Total	446,646,000	2,732,750	57,400,000	16,718,000	18,800,000	76,200,000		

* Consumption as cited by Ottomar Haupt, L'Histoire Monétaire de Notre Temps, Paris, pp. 21, 22.

† According to the census of Switzerland of 1870, the annual production of watches in that country for that year was 1,600,000, representing a total value of 88,000,000 francs. 37,969 persons were reported to be engaged in the business of watch-making in the four cantons of Switzerland famous for that industry.—*Larousse Dictionnaire Universel*, vol. xiv., p. 1221.

CONSUMPTION OF THE PRECIOUS METALS IN THE ARTS BY THE VARIOUS NATIONS OF THE WORLD.

Countries	GOLD.					SILVER.				
	Gross Coonsumption of Gold.		Deduction for old material used over.	Net Consumption of Gold.		Gross Consumption of Silver.		Deduction for old material employed.	Net Consumption of Silver.	
	Ounces.	Value. Dols.	Per cent.	Ounces.	Value. Dols.	Ounces.	Value. (1.2929 Dol.) Dols.	Per cent	Ounces.	Value. (1.2920 Dol.) Dols.
United States	482,250	9,969,000	10	434,025.00	8,972,100	3,858,000	4,967,200	15	3,279,300.00	4,239,120
Great Britain	643,000	13,292,000	15	546,550.00	11,298,200	2,893,500	3,740,400	20	2,314,800.00	2,992,320
France	675,150	13,956,000	20	540,120.00	11,165,280	3,215,000	4,156,000	25	2,411,250.00	3,117,000
Germany	472,605	9,769,620	20	378,084.00	7,815,696	3,215,000	4,156,000	25	2,411,250.00	3,117,000
Switzerland	482,250	9,969,000	25	361,687.50	7,476,750	1,028,000	1,329,920	25	771,600.00	997,440
Austria-Hungary	93,235	1,927,340	15	79,249.75	1,638,239	1,286,000	1,662,400	29	1,028,800.00	1,329,920
Italy	192,900	3,987,600	25	144,675.00	2,990,700	803,750	1,039,000	25	610,850.00	789,640
Russia	96,450	1,993,800	20	77,160.00	1,595,040	1,286,000	1,662,400	20	1,028,800.00	1,329,920
Above countries together	3,137,840	64,864,960	..	2,561,551.25	52,952,005	17,586,050	22,733,320	..	13,856,650.00	17,912,360
All other civilized countries	160,750	3,323,000	20	124,600.00	2,658,400	1,703,950	2,202,680	..	1,286,000.00	1,662,400
Total	3,298,590	68,187,960	..	2,689,366.25	55,610,405	19,290,000	24,936,000	..	15,142,650.00	19,574,760

The preceding Table, converted from one by Dr. Adolph Soetbeer,* embraces the result of his examination into the subject of the consumption of the precious metals in the arts by the various nations of the world, presumably for 1880.

The following Table gives the value and character of the gold and silver used in the United States in the arts and manufactures during the calendar year 1885, as reported by the persons and firms addressed.†

GOLD.

Manufacturers.	Number Addressed.	Replied.	Manufacturing.	United States Coin.	Stamped United States or Refined Bars.	Foreign Coin.	Old Jewellery, Plate, and other Old Materials.	Native Grains, Nuggets, &c.	Wire or Rolled Plate.	Total.
				dols.	dols.	dols.	dols.	dols.	dols.	dols.
Chemicals . .	341	219	39	32,040	13,903	..	6,063	29	4,341	56,376
Platers . . .	634	348	226	257,741	218,831	801	178,510	24,295	15,537	695,715
Gold-pen manufacturers . . .	34	22	11	7,433	34,886	2,867	990	3,526	6,753	56,455
Gold and silver leaf	72	51	46	58,150	527,453	2,000	31,050	19,700	39,001	677,354
Dental and surgical instruments. .	154	98	47	3,970	149,186	100	14,942	2,400	4,188	174,786
Spectacles and opticals , . .	383	217	79	52,707	62,420	642	16,269	314	2,291	134,643
Miscellaneous . .	106	73	27	116,604	44,168	8,000	17,337	1,000	3,835	190,944
Jewellery & watches	6330	3352	2232	2,298,733	5,183,187	164,503	582,554	451,629	485,241	9,165,847
Total . .	8054	4380	2707	2,827,378	6,234,034	178,913	847,715	502,893	561,187	11,152,120

SILVER.

Chemicals . .	..	..	..	91	305,165	..	73,561	106	2,165	381,088
Platers . . .	..	..	..	32,824	1,990,587	25,434	43,191	12,798	157,922	2,262,756
Gold-pen manufacturers . . .	..	..	..	55	3,191	..	249	558	5	4,058
Gold and silver leaf	..	..	..	..	21,881	..	708	20	23,512	46,121
Dental and surgical instruments . .	..	..	..	4,682	107,717	1,401	7,057	4,450	2,494	127,801
Spectacles and opticals . . .	..	..	..	2,587	42,424	155	2,750	210	942	49,068
Miscellaneous . .	..	..	..	838	5,330	..	268	70	1,017	7,523
Jewellery & watches	..	..	..	92,567	1,360,308	35,718	117,629	85,060	28,716	1,719,998
Total . .	..	..	..	133,644	3,836,603	62,708	245,413	103,272	216,773	4,598,413

* Verwendung des Goldes und Silbers, Jena, 1881.

† Report of the Director of the United States Mint for 1886, p. 56.

CHARACTER AND VALUE OF THE PRECIOUS METALS REPORTED BY MANUFACTURERS, JEWELLERS, AND OTHERS, USED BY THEM DURING THE CALENDAR YEAR 1885.

Character.	Gold.	Silver.	Total.
	$	$	$
United States coins	2,827,378	133,644	2,961,022
Stamped United States or refinery bars	6,234,034	3,836,603	10,070,637
Foreign coin	178,913	62,708	241,621
Old jewellery, plate, and other old materials	847,715	245,413	1,093,128
Native grains, nuggets, &c.	502,893	103,272	606,165
Wire or rolled plate	561,187	216,773	777,960
Total	11,152,120	4,598,413	15,750,533

CONSUMPTION OF GOLD IN THE INDUSTRIAL ARTS IN THE UNITED STATES.*

Fiscal Year.	Estimate of Director of the Mint, consumption of United States Gold Coin.	Estimate by Mr. Muhleman of consumption of Gold Coin and Bullion.
	$	$
1874	30,000,000†	12,600,000
1875		12,900,000
1876		13,900,000
1877		15,300,000
1878		15,200,000
1879		15,900,000
1880		16,700,000
1881	3,300,000	17,800,000
1882	2,700,000	18,000,000
1883	2,500,000	17,500,000
1884	4,875,000	17,000,000
1885	5,000,000	15,600,000
Total	48,375,000	188,400,000

* Production of the Precious Metals in the U.S., 1886, p. 81.

† Includes short estimates for 1881, 1882, and 1883.

ESTIMATE OF CONSUMPTION OF THE PRECIOUS METALS IN THE INDUSTRIAL ARTS FROM JULY 1, 1873, TO DECEMBER 31, 1885.

	Gold.	Silver.
	$	$
New bullion (exclusive of old jewellery, plate, &c., used over) based on reports of manufacturers for 1883, twelve and a half years . .	107,875,000	66,000,000
United States coins, sum of the annual estimates of Director of the Mint:		
Gold	48,375,000	...
Silver, say, $200,000 per annum . .	...	2,500,000
Total	156,250,000	62,500,000

The following Table gives the industrial consumption of precious metals in the United States for the years 1880, 1881, 1883, 1885.*

CHARACTER OF MATERIAL USED.	1880.		1881.		1883.		1885.	
	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.
United States coins . .	2,408,768	541,834	3,315,882	72,190	4,875,587	216,637	2,827,378	133,644
Stamped U.S. or refinery bars	5,511,047	2,749,190	6,171,317	3,127,432	7,137,661	4,552,172	6,234,034	3,836,603
Foreign coin	714,378	173,145	599,524	188,799	194,400	154,273	178,913	62,708
Old jewellery, plate, and other old materials . . .					876,641	221,951	847,715	245,413
Native grains, nuggets, &c. .	..	..	..	..	702,387	71,557	502,893	103,272
Wire or rolled plate . .	..	..	..	..	672,688	339,940	561,187	216,773
Total	8,634,193	3,464,189	10,086,723	3,388,421	14,459,464	5,556,530	11,152,120	4,598,413

The result of the inquiries for the years 1881 and 1885 are practically the same, while that of 1880 is but little less, a smaller number of consumers having reported. In 1883 the sum reported was of gold over $4,000,000 more, and of silver $2,000,000. From the results of these inquiries it appears that the consumption of United States gold coin in the arts does not at present exceed $3,500,000 per annum.

* Report of the Director of the U.S. Mint, 1886, p. 57.

STATEMENT SHOWING THE VALUE AND CHARACTER OF THE GOLD AND SILVER USED IN THE ARTS AND MANUFACTURES DURING THE CALENDAR YEAR 1883.*

MANUFACTURES.	Number Manufacturing.	GOLD.							SILVER.							Total Silver and Gold.
		United States Coin.	Stamped United States or Refinery Bars.	Old Jewellery, Plate, and other Old Material.	Foreign Coin.	Native Grains, Nuggets, &c.	Wire or Rolled Plate.	Total Gold.	United States Coin.	Stamped United States or Refinery Bars.	Old Jewellery, Plate, and other Old Material.	Foreign Coin.	Native Grains, Nuggets, &c.	Wire or Rolled Plate.	Total Silver.	
		Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.	Dols.
Watch-cases	32	575,812	2,976,550	38,101	1,508	520	5,817	3,598,308	35,200	1,777,193	31,937	219	1,000	50	1,845,599	5,443,907
Watch-chains	14	374,997	286,884	1,907	600	135,410	27,202	827,000	524	14,768	..	..	6,790	1,462	23,544	850,544
Dental Supplies	7	700	33,437	3,775	..	..	..	37,912	450	6,060	..	..	..	228	6,738	44,650
Pens	14	14,578	90,325	6,100	5,227	2,134	27,560	145,924	216	4,254	100	1,655	505	..	6,730	152,654
Instruments	45	68	..	3,568	..	621	942	5,199	931	3,752	693	755	864	6,995	13,990	19,189
Leaf	51	178,424	792,551	57,498	6,816	6,700	42,835	1,084,824	11	22,697	4,107	300	835	18,933	46,883	1,131,707
Plate	219	379,291	67,928	5,500	590	8,933	66,626	528,868	16,856	1,710,515	40,761	7,690	8,495	281,977	2,066,294	2,595,162
Spectacles	41	192,400	7,169	8,830	1,315	4,987	727	215,428	3,631	16,461	1,254	205	250	1,981	23,782	239,210
Chemicals	27	7,438	7,685	3,551	550	207	12,180	31,611	9	375,429	35,554	500	1,580	3,347	416,419	448,030
Jewellery and watchmakers' supplies	11	24,498	13,963	9,123	..	1,569	30,054	79,227	245	4,806	800	..	1,505	975	8,331	87,558
Jewellery and watches	2,273	3,125,738	2,861,149	738,688	177,794	541,306	458,745	7,905,163	158,564	616,237	106,745	142,949	49,733	23,992	1,098,220	9,003,383
Total	2,734	4,875,587	7,137,661	876,641	194,400	702,387	672,688	14,459,464	216,637	4,552,172	221,951	154,273	71,557	339,940	5,556,530	20,015,994

* Production of the Precious Metals in the U.S. for 1885, p. 48.

No one has ever been able to explain what becomes of all the gold and silver produced in the world.* Mr. M. S. Mulhall† has given some interesting statistics relating to it. According to him the total annual losses of the precious metals from shipwreck, fire, wear, tear, and abrasion of coin, amounts annually to about 2 tons, or £280,000 or $1,400,000. It has been estimated that gold coin loses 2 per cent. in 100 years, which is £147,000 or $505,000 per annum, on a production of £736,000,000 or $3,680,000,000. The loss by shipwreck is estimated as $\frac{1}{6}$ of the ratio of loss of all sea-borne merchandise, or £2 or $10 on every £1,000 or $5,000 shipped. The quantity of sea-borne gold for 1871 to 1880 averaged £50,400,000 or $252,000,000; the loss by shipwreck would therefore be £101,000 or $505,000. If the loss by fires is estimated at £32,000 or $160,000

	£	$
Loss by abrasion, 2½ per cent. in 100 years, or per annum	147,000	735,000
Loss by shipwreck, $\frac{1}{6}$ of ratio of loss of other sea-borne merchandise	101,000	505,000
Loss by fires	32,000	160,000
	£290,000	$1,400,000

McCulloch estimated the consumption for jewellery as £4,000,000 or $20,000,000 a year. The uncoined reserve was better understood eighty-four years ago than now,‡ when it was said the precious metals have a twofold use, for manufactures and for coin. If there is a superabundance of coin it will be melted and manufactured. If there is a deficiency of coin the plate will be melted and coined.

* Vol. i., p. 52. † Contemporary Review, 1885.

‡ Edinburgh Review, 1803.

CHAPTER VIII.

TREATMENT OF MERCURY IN CALIFORNIA.

THE ores of mercury which are found in California are metallic mercury and cinnabar. They occur in large quantities, and both of them are mined and treated as ores. Minerals of mercury occur occasionally, such as the selenides, but they are rarely found, and have no metallurgical interest. Cinnabar, the principal ore of mercury, was found in California many years before it was worked. There was no special use for it, and no regular records of its working were kept previous to 1850, when the development of the gold mines caused such a demand for it that it was worth $114.50 per flask or $1.50 per pound. From that time until 1866 it gradually fell to $45.90, and remained so until 1870, when it rose to $118.50 for two years, and then steadily declined until 1883 it fell to $26. In 1888 it fluctuated from $37 to $48, the average for that year being $42.50. In 1878 there were 30 producing mines; in 1888, owing to the low price of quicksilver, there were only 10.

Cinnabar in California is generally found near the junction of serpentine and sandstone; sometimes in the serpentine, as at the Redington; sometimes in the sandstone or nearer it than the serpentine, as at the Manhattan; and sometimes exclusively in the sandstone, as at St. John's Mine near Vallejo, on the Oat Hills Napa County, the Oceanic in St. Louis Obispo County, and the Altoona in Trinity County, which is the most northern mine of the state. Sometimes ore is found in clay, as at the Sunderland Mine, and upon Mount Shasta, where it occurs in seams of clay. These mines are situated so high on the mountain that they cannot work more than four or five months in the year, as the snow is very deep. At the Geyser, New Almaden, Guadaloupe, and the outcrop of the Oakland mines, the

serpentine is very much decomposed on the outcrop. The decomposition goes so far into the rock that it is only in the large pieces that its real nature can be seen. This decomposition is going on still in all the mines in the serpentine, the waters from which, and from the surrounding country, contain large quantities of magnesium sulphate. The sandstone is sometimes very fine and soft, and forms what is called the mud rock at the Manhattan, American, and New Idria mines. At the Oceanic Mine the sandstone is so crumbly that it fell easily to powder on being worked, so that nearly all the ore had to be made up into adobes, to which mud had to be added to hold it together. Where the sandstone is not decomposed, it very often contains considerable quantities of petroleum, which render the extraction of the mercury more difficult by increasing the amount of soot.

These ores, especially those in the sandstones, are very often found associated with chalcedony, as at the Manhattan, Missouri, and Kentuck mines, where in portions of the mine the rock is chalcedony, more or less impregnated with mercury. But these masses are very irregular in the mine. The rock is sometimes rich enough to yield 3 to 10 per cent. and sometimes more, of mercury. A microscopic examination of this chalcedony containing cinnabar shows that the cinnabar is crystallised, the crystals being in a great many cases doubly terminated, though to the eye nothing but a red stain is perceptible.

The deposit containing the ores of North California makes its appearance at Vallejo, where it has been opened in several places. The greatest development, however, was further North. The largest number of mines were formerly situated in Sonoma Lake and Napa counties, and this district was very extensively worked while the price of mercury was high. In South California it is comprised within Santa Clara and Fresno counties. As the serpentine rock is very susceptible of decomposition, it has often been thoroughly decomposed on its outcrop to a very considerable depth. Where the serpentine has formed cliffs it has been washed into the valleys, so that there is then a large deposit of material from the decomposed rocks. In such cases it will be found that what appears to be nothing more than ordinary dirt will contain frequently from 2 to 3 per cent. of metallic mercury, with only a

trace of cinnabar; in which case, as at the Sonoma Mine, it was made up into adobes and distilled. Not unfrequently the outcrop of the rock where it is not decomposed is filled with metallic mercury, so that by striking a pick into the rock, a pound or more of the metal will sometimes spurt out at a time. This was particularly the case at the Rattlesnake Mine, which was in serpentine, and also at the Wall-street Mine, where the rock is silicious; but where it was sometimes so thoroughly impregnated with mercury that from a hand specimen several ounces could be detached by slight jars. Such rock as this was found in several localities in every stage of impregnation, from mere microscopic globules up to those yielding large quantities of mercury which make very rich ore; there is, however, not very much of it in any one place. It consequently cannot be depended upon, as it is only found in the first workings. The ore in depth is always cinnabar.

The ore is almost invariably associated with marcasite, whose constant decomposition often produces fine sharp fibres of epsomite, which hang from the roof of the mine workings in bunches as large as a bushel basket. At the Redington Mine metacinabarite was at one time quite frequently found. Bitumen, both in solid and liquid form, is also frequently found in pockets. At New Almaden lumps of apparently pure cinnabar are occasionally found which give a voluminous residue of pulverulent carbonaceous material when distilled without access of air. This would seem to show that the bituminous substance, possibly idrialite, is intimately associated with the cinnabar. In some places in this mine bitumen, both liquid and solid, is associated with the ore, sometimes impregnating the rock, sometimes covering it and the crystals of dolomite with which it is associated. It is frequently spotted with globules of metallic mercury. At the Sulphur Bank Mine in Lake County, the ore is found in an extinct geyser, which still produces boiling water, associated with very large quantities of gypsum, alkaline borates, and sulphur. It is found directly on the shore of the lake. The average yield of the ore is not over 1.75 per cent. None of these ores are of themselves difficult to treat. At the Sulphur Bank, however, there is sometimes so much of the alkaline borates that

if the temperature is raised high enough to drive off all the mercury, the ore is likely to frit and make pasty masses, which hang in the furnace. When the fine ore is charged, or when it becomes fine from decripitation of the large lumps in the furnace, it will run when it is red hot, like many other fine ores, almost like water. But these difficulties have been mostly overcome by the use of the shelf furnaces. Sometimes there is more sulphur than cinnabar in the ore, which is a detriment to both sulphur and cinnabar, making the sulphur impure, and rendering the cinnabar difficult to work on account of the soot. In some of the early constructed furnaces the accumulation of soot from the excess of sulphur has been known to penetrate as far as the fan blower at the end of the furnace condensers, and completely prevent its revolution. In order to get rid of the inconvenience of this accumulation of soot, as well as to get a commercial value from the sulphur, it is now separated by steam and sold. It is claimed that the sulphur alone pays all the expenses of working the mine. The mine was first worked for sulphur, and the cinnabar which it contained was considered to be an inconvenience, and was carefully picked out and thrown away, as its richness and value were not known. This ore is now treated, and forms one of the largest mercury mines in the State. The deposit is very large and very rich, but the mine will always be difficult to work, as the ore occurs in the vicinity of boiling mineral springs, which discharge at the same time very large quantities of carbonic acid gas. The ore is mined in an open cut, and in several places the lake level has been reached. Besides the rich ore there is a very large quantity of poorer ore yielding 1 or 2 per cent. which the sulphur makes almost impossible to work. As the sulphur is in such large quantities, it is very necessary that the work of mining should be done without fire, the use of which is strictly forbidden to the workmen.

The ore is divided into two separate categories, that containing a large and that containing a small quantity of sulphur. That containing a large amount is first treated for sulphur in a steam furnace, Fig. 120. The upper part B, which is round, is made of thick boiler plate bolted on to a cast-iron bottom C. The charging door A is hinged, and has a handle with which it is

screwed down. The lower part is provided with a heavy cast-iron grate D, on which the ore rests. The ore is charged upon the grate and the furnace closed. Steam at the temperature of 230 deg., and at a pressure of about 50 lb., is introduced through the pipe G; the sulphur is melted, liquates, and runs through the grate into the lower part, from which it is discharged by means of the stop-cock F, which is surrounded by a steam jacket to keep it hot. When no more sulphur flows the discharge door E is opened, and the residue is treated for mercury.

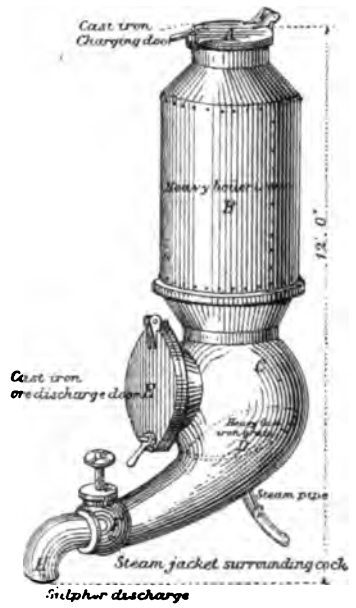


FIG. 120.—FURNACE FOR SEPARATING SULPHUR FROM MERCURY ORES.

The attempt to concentrate the poor ore at the mines has been made with more or less success with various kinds of machines. The result of these concentrations was treated in retorts, of which there formerly were fifteen or twenty. The fine ore was treated in a furnace somewhat similar to the old Idria furnace, but the ore was made to fall over triangular shelves made of tile, and arranged somewhat similar to the Gerstenhoffer furnace. The condensers were arranged as in the old Idria furnace, although it is continuous. The furnace treats 12 to 13 tons in twenty-four hours.

The metallurgy of mercury in California is essentially different

from that employed in all other countries in this respect, that all the furnaces use fans placed beyond the condensing apparatus, to furnish their air, which not only give an absolute control of the draught, avoid the necessity of employing chimneys, but prevent the action of the fumes on the surrounding life and vegetation. The consequence of this arrangement is, that the draught is always towards the interior of the furnace or condensing apparatus, so that if a crack appears, or a door is left open, no fumes will escape; cases of salivation are consequently very rare. The draught can always be regulated to suit the working of the furnace, or the irregularities of the weather, by changing the velocity of the fan. In all of these methods the furnace itself is of very much less conseqence than the condensing apparatus, though it should always aim to extract the metal from the ore at the lowest possible temperature, and to be able to work the ore after the least possible mechanical preparation. It is consequently on the condensing apparatus, rather than on the furnace itself, that the greatest number of experiments have been made. The furnaces used are intermittent and continuous in their action, and of these last, the one which has the best condensing apparatus will be the best. It might be said that almost the whole metallurgy of mercury consists of condensation, for which an abundant water supply is necessary.

Generally the ore, as it comes from any of the mines, is more or less hand-picked. The attempts which have been made to treat the ores mechanically have usually not been successful, as cinnabar is so firable that more floats off with the water than is concentrated; the most successful machine has been the Frue vanner. As a general thing the apparatus used for this purpose has not been well managed, as there is a large amount of cinnabar still in the tailings. In some of these mines all that is saved is a matter of clear gain; but for the fact of the concentration the whole would be lost, whereas with it, only a very considerable percentage is. It is doubtful whether the concentration by any of the methods now in use, except a moderate amount of hand-picking, will be worth making. No concentration of the fine ore has as yet been successful. Usually no attempt is made to concentrate the ore. It is taken as it comes

from the mines, and formerly if it contained binding material the fines were made up into adobes; if it did not contain any, some was mixed with it. The adobes had no regular size or weight, and were treated in the furnace as large ore. The making of adobes has now been abandoned. The ore is only sized and is treated in special furnaces adapted to each size without further preparation.

The processes by which the ore is treated are, first, the process of Distillation; second, the process of Precipitation; third, that of Roasting. Distillation is used when the mercury of the outcrops is found in sufficiently large quantities to warrant the ore being treated by this method alone. This, however, is not often the case. It has only very rarely been used as a separate process. It consists simply in driving off the metallic mercury by heat and condensing it. The distillation is usually done in shaft furnaces, which are continuous in their action. It was quite extensively used for a short period in North California on outcrop ore, and does not require any special process or furnace. Such ore is charged in the furnace either as regular ore or as adobes. As native mercury is only rarely found and not in large quantities it is usually mixed with the other ores and treated with them. Precipitation is done in retorts with lime, and consists of admitting oxygen enough, by means of an excess of air, to produce calcium sulphate and free murcury. Roasting is either done in continuous or non-continuous furnaces, which are either retorts, or various kinds of shaft furnaces. The reaction, as the ore always contains more or less moisture, consists in volatilising the sulphur, and oxidising it so as to produce sulphuric acid, which is taken up with the moisture, and runs to waste, while the mercury becomes free, and is caught in condensers The furnaces, which are not continuous, are a modification of the old Idria furnace which was used at the Redington mines, and is still used at the New Almaden mines in Southern California. Those which are continuous are the Luckhart, which was used at the Sonoma, Rattlesnake, Manhattan, and California mines, and elsewhere. Livermore's furnace, adapted only for treating fine ores, has been successfully used at the Redington Mine, and at other places. The Exeli and the Hüttner and Scott furnaces are used at New Almaden and elsewhere. At the Abbot Mine, in Colusa County, an ordinary

reverberatory furnace, with a slightly inclined hearth, was used. The ore had to be worked in the furnace. The work was well done, but the method was expensive.

There are a large number of varieties of shaft furnaces which are continuous, some of which work with water jackets, which appear and disappear, according as the price of mercury rises or falls, and are usually imitations of others which have been more or less successful, for there is a desire in California, as elsewhere, to improvise principles as well as methods, while the persons attempting to improvise are very frequently without instruction, and without experience in metallurgy. It is not unusual that when new mines are discovered they are made to work for a little while on selected ore. Then new furnaces are erected more or less in defiance of principles, but "the future of a great enterprise" is predicted on the results obtained, which are made to show a large profit, but usually the new methods disappear, and the new mines are closed on account of "hard times," to the great financial embarrassment of those who have invested in them.

The only effort made to sort the ore is a rough attempt at hand-picking. It is generally sorted into approximate sizes, over rudely-constructed gratings, the object of which is to separate the fine ore as much as possible, since most of the furnaces would be clogged if fine ore in any considerable proportion were allowed to enter them.

While the price of mercury was very high no assays of any kind were made at the mine. The superintendent of the works or the mines judged by "experience" that the ore contained 1, 2, 3, 10, or 20 per cent., as the case might be. There was consequently very little dependence to be placed upon any of the statements of the advocates of the different kinds of furnaces that their furnaces actually yielded a higher percentage than those of their neighbours, or, in fact, that they yielded any given percentage at all. The only statement that could be relied upon was, that they produce in twenty-four hours a given number of pounds of mercury. All attempts at concentration at these works, except a moderate amount of hand-picking, have failed. No experiments in concentrating the tierras, as the fine ore is called, has ever succeeded.

The mercury is delivered to the market in iron flasks, which contain 76½ lb. each. It is accurately weighed in a pan-scale constructed for the purpose, and poured into the flasks. They are closed with an iron screw for a cork, which is screwed tight with a long lever, and then shipped to the consumer. When the flask contains more or less than this, it is called an irregular flask. These are becoming less and less common, and are likely to go out of date altogether. Foreign quicksilver is sold in flasks containing only 75 Spanish pounds, which is equal to 76.07 avoirdupois. American flasks are well made, and at second-hand are worth $0.75. Foreign flasks are undersized and badly made, and are worth only about $0.12.

In the year 1876 the State of California produced 75,000 flasks, of which nearly 70 per cent. were exported to China. Only 26,000 were used in California and Nevada. In 1886 29,981 were produced, of which 16,301 flasks were exported. In 1887 33,760 flasks were produced, of which 22,119 were exported.

The processes which are used for the treatment of ores of mercury are:—

I. Distillation.
II. Precipitation.
III. Roasting and Reduction. { Intermittent furnaces. / Continuous furnaces.

I. Method of Distillation.

This method can only be used when the ores contain free mercury. There were formerly a few mines of this character in which the mercury occurred scattered through a porous rock, and other works in which the metal was contained in the earth near the outcrop of the decomposed serpentines. The rock was usually treated as other ore, the earthy material was made up into adobes and either treated by itself or with large ore in some kind of a continuous shaft furnace, as in the Luckhart furnaces at the Sonoma. Neither the treatment of the ore nor that of the soot, differs essentially from that of the continuous processes generally, except that the difficulties and inconveniences connected with the presence of sulphur and its oxidized products did not have to be contended with. The process was used rarely, and only during the times when the price of mercury was very high. The

same furnaces and condensers were used as in the process of roasting. It should be considered rather as a method of working a "prospect" than as a regular process, as in any event if a mine was developed out of the "prospect" the ore would turn into cinnabar.

II. Method by Precipitation.

The method by precipitation is the oldest, the simplest, and the worst of all the methods used in extracting mercury. It is based on the principle of having some substance such as lime or iron to take up the sulphur in the ore, and set the mercury free. Its use involves so much expense, loss of mercury, and danger of salivation, that it has been abandoned in California, although it was at one time largely in use at the small mines, and was very much perfected. In its very best form, however, it is expensive and unsatisfactory. The first attempt to reduce cinnabar in California was made in the cast-iron boilers in which whalers tried out whale oil. Iron covers were luted on to them, but this was so imperfectly done, and so much salivation resulted, that they had quickly to be abandoned. Iron retorts set in brickwork, with a more or less perfect system of condensation, generally consisting of iron pipes, cooled with water, were quickly substituted for them; these were very extensively used while the price of mercury was high, both as prospecting tools and for regular working. They were built as single retorts or in regular batteries, but have long since been abandoned.

At the Missouri Mine, near Pine Flat, the ore yielded about 2 per cent.; it was cinnabar scattered through chalcedony, which was found in very irregular masses in serpentine. The pieces were reduced to about one cubic inch in a Blake's crusher, and were then treated in retorts, of which there were two benches which contained respectively 200 lb. and 350 lb. at a time; so that the small ones treated 1000 and the large ones 2100 in twenty-four hours. They were ∩-shaped, 18 in. wide, 12 in. high, and 9 ft. long. They were charged every twelve hours by means of a shovel. It took three quarters of an hour to charge and discharge the large retorts, and half an hour for the small ones. They were never full, as there would have been danger that the pressure of the decomposing ore might force the luted joints, so

that the vapours of mercury would have been discharged into the works. At the Lost Ledge Mine the same company had three retorts which were 5 ft. long, with a capacity of 160 lb., and consequently could treat 3000 lb. of ore in twenty-four hours. As the wear and tear is the same, and the cost of erection is not very much less, it was very disadvantageous to use small retorts.

When the furnace is ready for charging, the workmen tie wet sponges over their mouths and noses, and then open the door. Before the door is opened a little fan placed on the condensing pipe is set in motion, which causes an aspiration through the retort, so that the mercurial vapours are drawn into the condensing chambers. As soon as the back of the retort can be seen, the ore is drawn with long hoes to the front of the furnace. The drawn ore is at a cherry-red heat, and as it is not entirely free from mercury, a considerable quantity of fumes are given off, so that the precaution of wearing the sponge is indispensable. The ore was carried as quickly as possible a short distance from the furnace, and extinguished with water, and a new charge introduced. When water was scarce, only small quantities at a time were thrown on to the ore, which seemed rather to increase than decrease the mercurial fumes. When it became very scarce, no water at all was used. The men are cautious about being near the pile of spent ore any longer than they can help, for, as the ore is very hot, steam, which carries mercurial vapours with it, is given off from it for some time. When no water is used, and the pile is left to itself, it smokes for some time. The men are very rarely salivated, however, though they run a great risk. The small retorts produce from 3½ lb. to 70 lb. of mercury, and the large ones from 10 lb. to 100 lb. in twenty-four hours. Each retort is connected with the condensers by a horizontal pipe 6 in. in diameter and 30 ft. long; large enough to insure its not being clogged with soot. They were formerly connected with the front of the retort, where they were always in the way of the workmen, and in danger of being damaged by the charge drawn from the furnace lying against them. They were afterwards made to connect with the back of the retort.

The condensing apparatus consisted of two cast-iron boxes 3 ft. by 2 ft., and 2 ft. high, turned down into cast-iron tanks

with an inclined bottom, which was covered with water, so as to make a water seal. Into the bottom of these tanks a wrought-iron syphon, with arms sufficiently long to counterbalance the pressure of the water, was placed so that the mercury fell from it continually into a cast-iron pan. The mercury commenced to flow two hours and a half after the charge was introduced. A small fan blower to suck the vapours from the retorts was placed between the two condensers. This fan was enclosed in a box 18 in. square, which had an opening 1 ft. in diameter, closed by hydraulic packing, so as to give easy access to the interior. The pipes and condensers were cleaned once a week.

One hundred and forty pounds of soot and mercury mixed were collected from the small furnace. The mercury is scattered through the soot in very small globules, and in order to separate it, it is put into a sheet-iron pan 8 ft. long, 3 ft. wide, and 6 in. deep, inclined about 25 deg., which is moved from place to place as convenience may require. The soot is placed at the upper end, and in order to separate the mercury which is free or in flour, it is worked with a hoe for a quarter of an hour. Three to five pounds of unslaked lime are then stirred into it. As the soot is damp the lime soon commences to slake; and the whole is left until it is entirely slaked. It is again worked with a hoe for three hours. About one barrel of lime, which costs $3, is used per week for treating the soot. The mercury, as it collects, runs into the lower part of the pan, and is caught in a pot. The residue, which has twice the bulk of the original soot, is charged in the retort as ore; such a charge will produce 10 lb. more mercury than usual. From 140 lb. of soot 120 lb. of mercury are obtained directly, and about 10 lb. more from the soot charged in the furnace. When there is only a small amount of soot to be treated, an ordinary iron kitchen sink, 3 ft. by 18 in., is used. The drain hole is placed over an enamelled iron vessel 1 ft. in diameter, and 6 in. deep. This vessel is used only because ordinary kitchen utensils are cheaper than the special apparatus, which would work no better.

As the two benches of retorts were not near each other, each one required one man per shift of twenty-four hours, who was paid $3.25. They burned three and a half cords of wood in twenty-four

hours, which costs about $6 a cord. The usual price for labour was $64 a month with board.

The Expenses for Twenty-Four Hours.

	$
Two men at the retort	6.50
Three and a half cords wood at $6.00	21.00
Lime for soot	.50
Crushing ore two days at $1.25	2.50
	$30.50

At the American Mine near Pine Flat all the ore previous to 1875 was crushed fine, in a ten-stamp California mill run at a high velocity. The use of such a stamp on cinnabar, which is exceedingly friable, and on an ore which is very lean, can hardly be excused on the ground that the works were erected by a California gold miner. The stamped material was mixed with lime and treated in retorts. The average yield of the ore was said to be 2 per cent. The retorts were made of cast iron, ⊓ - shaped like an ordinary gas retort, and arranged in benches of three, the centre one being the highest. There were twelve of these retorts, 9 ft. long, 2 ft. wide, and 18 in. high in the middle, for which there were four fireplaces. Ten per cent. of quicklime was used for a charge of 150 lb. of crushed ore. The ore and lime were mixed together in an iron trough, 6 ft. long, 9 in. wide, and 6 in. deep. This was carried by three workmen, one of whom held at the end an iron handle 2 ft. long, with a crossbar of wood of the same length, while the other two supported the middle of the trough on an iron bar curved in the centre to fit it. The trough was introduced into the furnace, and pushed up to its further end. It was drawn out by the man who held the end handle by short jerks, so as to leave the charge distributed over the bottom of the retort. The doors were then fastened by means of thumbscrews, and made tight with wood ashes, and fired during four hours. The capacity of the retort was 500 lb. of ore in twelve hours. The condensation pipes were fitted with flanges to the back of the retort. A pipe about 18 in. long connected it with a vertical pipe 5 ft. to 6 ft. long, which fitted into a horizontal pipe 6 in. in diameter, placed in a long tank of water below the level of the front of the furnace. Every six retorts discharged their condensed mercury through a syphon

pipe. The soot was treated with lime. If the proper quantity of lime had been added to the ore, there would have been much less soot to treat.

It was surprising to see works which were new in 1874, and carefully built, and upon which no money seems to have been spared, put up to treat such very small quantities of a very lean ore by such a system. Economy of construction or of management did not seem to have been taken into account. The air around the works for some distance was filled with mercurial vapours, and though the men wore wet sponges over their mouths and noses, most of them were more or less salivated. I did not visit the mines, but any mine which would justify such expensively constructed works would certainly justify the erection of less wasteful furnaces. It is not surprising to learn that the retorts were very soon ruined and the furnaces torn down. A shaft furnace was then erected which was afterwards modified by having the Knox fireplace put into it, which treated 10 tons to 15 tons a day. These works have long since been abandoned.

The reason why the retort system, which is neither economical in running or repairs, and has no advantage in yield, was formerly so often found in use, is that certain encyclopædias, works on chemistry, and other books which are in general vogue, have copied, and are still copying, experiments made fifty years ago as giving some of the best results ever obtained in the metallurgy of mercury, which might perhaps have been true at that time. Most of these books state that the retort system is still in use in California, and especially at New Almaden. They were, however, abandoned in those works for the treatment of ore about 1850. Up to 1860 they were used for making assays, but since that time have been abandoned altogether. As these books are the only literature on the metallurgy of mercury to which the miners generally have access, the retort is usually found in most places where the metallurgy of mercury is in its infancy. There is perhaps an economy in the first installation, and this, with companies of small capital, seems to be a great desideratum. With the high prices of quicksilver which ruled from 1873 to 1875, every place where ores were found was worked.

As most of the deposits were irregular and uncertain, the owners of such mines did not feel themselves justified in going to any great expense, until the opening of the deposit showed that there was a future in it, and for that reason used retorts as the cheapest way to work what ores they had. I saw no well-developed property, nor indeed any property which appeared to have a future, which was worked with retorts except in just such instances as these, where the retorts were used to bring in an income while the property was being developed, or where the mines were evidently being worked in good faith with too small a capital, or in order to have some production so as to be quoted on the stock list. Since the fall in the price of mercury they have been abandoned altogether, for sanitary reasons if for no other. Their use, on a large scale, ought to be prohibited by law.

III. Method of Roasting.

The method of roasting is the process which is the most extensively used. A very large number of furnaces have been invented for its application, most of which have come into use in a few works while the prices of quicksilver was high, and in a few works only, and have gone out of use when owing to lack of ore, or lack of profit, the mines closed. The use of the retort necessitated the crushing of the ores in order to mix them intimately with the lime. It required such an expensive plant for crushing that only very rich and selected ores could be used, and necessitated the throwing away of so much of the product of the mine that many attempts were made to crush and then concentrate the ores. This necessitated the additional expense of drying them. The choice was between rejecting all but rich ores, or of crushing, concentrating, and drying all the ores. No experiments in concentration, further than rough hand-picking, have been successful, and if they had been, the extremely limited capacity of the retort, and the expenditure of so large an amount of capital for a small output, and the constant salivation of the men, would have prevented its extended use. The first attempts to improve were made in California previous to 1850 by introducing the methods then in use at Almaden in Spain, and Idria in Austria. The first furnaces, whether built of brick or of sand-

stone, were so badly constructed, and so much of the metal was absorbed by the walls or lost in the foundations, that it was frequently profitable to excavate the earth from the sides of the furnace, and treat it as an ore. The amount of it in the masonry of the furnace, and in the ground directly beneath it was sometimes sufficient, when such material was treated as an ore, to pay the expenses of tearing down and of reconstructing the furnaces. In the year 1863, at New Almaden, 2000 flasks were taken from the ground under or near one of the furnaces. The mercury had penetrated to the rock at a depth of 27 ft. In 1874 they worked the ground from the front of the furnace, extracting it in a pit 12 ft. square and 25 ft. deep.

These furnaces were, however, gradually improved both in form and construction, and one of them, No. 6, Figs. 123 and 124, was still in use at New Almaden in the summer of 1889. The great difference between this furnace and those used in Austria and Spain is that the ore is no longer supported on arches as at Idria, which being constantly exposed to the flames of the fireplace below, and to the reverberated heat above, rendered the furnace expensive both in construction and repair. It has the fault, however, common to all such furnaces, that it is not continuous, and that the walls as it was formerly constructed were more or less porous, and sometimes even disintegrated from the absorption of such large quantities of mercury. The walls, beside, were constantly cracking, and it was impossible to prevent salivation both while the furnace was working and while it was being discharged. Besides this, in every mercury mine the large ore is but a small part of the product, and the fines had either to be made into adobes, before they could be treated or added in very small quantities, or to be thrown away, thus adding to the expense. This led to improvements which eventually culminated in the invention of continuous furnaces, but the difficulty of the fines still remained. The furnaces were finally improved so as to make them continuous for the treatment of all classes of large as well as of fine ores.

Modified Idria Furnace.—The Redington Works.

The practice of the Redington works with the old Idria furnace

was one of the best examples of non-continuous work in shaft furnaces. These were the largest works in Northern California, and for a long time had no rival in this district except, perhaps, the Sulphur Bank Mine. As everything relating to this variety of furnace, which is destined in future years to disappear altogether, is of historical interest, it has been fully described. Some details of the New Almaden furnace of the same general type, which is equally well managed, are also given. The most important question in the metallurgy of mercury is condensation, and it is of even more importance than the construction of the furnace. The two questions are, however, intimately connected, and are engaging the serious attention of all persons interested in the metallurgy of mercury.

The Redington Quicksilver Company own a large property or Hacienda, which has been called Knoxville, and a small village connected with their works is known as the Redington Mine. They formerly had at this mine two modified Idria furnaces, which have since been reconstructed into Livermore's fine ore furnaces These furnaces were situated within a few feet of the ore shaft. The mine formerly produced 700 to 800 tons of cinnabar per week. It occurs in serpentine. A very large quantity of high grade ore was found, but the average yield of the mine was about 3 per cent. Cinnabar and metacinnabarite are both found crystallised in considerable quantity. Epsomite resulting from the action of decomposing white iron pyrites on the serpentine is also found in very great abundance in acicular crystals over a foot long. The ore is very intimately associated with the pyrites. About one-tenth of it comes from an opening made in the side of the hill. From the mine the ore is dumped on to screens. two of which are placed one over the other. The upper screen is made of bars $1\frac{1}{2}$ in. in diameter, 2 in. apart at the top, $2\frac{1}{4}$ in. at the bottom. The screen is 8 ft. long, 5 ft. wide at the top, and $5\frac{1}{2}$ ft. wide at the bottom. What passes over this screen only is hand-picked. What passes through falls upon a wire screen of $\frac{3}{4}$ in. mesh. Most of what passes over this screen goes directly to the furnace, and is charged with hand-picked ore. What passes through it is treated as fine ore. All the large pieces are broken by hand. The fine ore was formerly made into

adobes, which were used in the modified Idria furnaces exclusively. No dirt is mixed with the adobes. The ore is so filled with easily decomposing iron pyrites that the action of the water effects a partial decomposition of the magnesian rock which binds the particles together.*

The adobes are made in wood frames roughly put together by the workmen, and intended to be 9 in. by 4 in. by 4 in. When such adobes are dry they weigh about 12 lb. They are sometimes made 12 in. by 5 in. by 5 in., when they weigh about 25 lb. which is an unusually large size. The smaller size is the one most generally used. Seven men at $2 per day can make 6000 adobes; each man can mould 1000, but six men require one helper. The adobes therefore cost about $2.33 per thousand. The actual cost is difficult to ascertain, for adobes are not like ordinary ore which can be dumped anywhere. They are really more tender than ordinary sun-dried brick. They are generally made during the dry season only, and they must then be carefully stored in sheds until wanted. They will therefore have to be handled at least three times after being made, once in storing, once in being removed to the furnace, and once in charging, which causes many of them to break up and become entirely reduced to powder and must be mixed with water to be made over again; broken in two or three pieces they still can be used, but there comes a limit when the pieces are too small which limit is reached all the more rapidly as the adobes contain less binding material. No account is usually taken of this loss; it has not been considered worth while to estimate it, and it will probably never be done, as adobes will no longer be made, as some of the best metallurgical ability in California is being expended on special furnaces for the treatment of fine ores. The cost of the adobes delivered at the furnace ready for charging, as near as can be estimated, including making loss by breakage, transporting, and storing them, is about $5 per thousand.

The modified Idria furnaces, Fig. 121, were built one on each side of the condensers E, which were at right angles to them. Each of these furnaces treated 100 tons of ore and 4000 to

* This effect is similar to that at Agordo.—*School of Mines Quarterly*, vol. ix., p. 132.

5000 adobes per week. They were built of a very porous sandstone found a short distance from the works. Each furnace had a double fireplace A, 20 in. wide, 17 ft. long, and 15 ft. high. The total inside height of the fireplace compartment was 20 ft. The ore chamber B had on both ends a brick screen with fourteen openings of the size of one brick, and two bricks apart, across the furnace, and fourteen openings in height, also two bricks apart. The fireplace screen had to be rebuilt once in seven

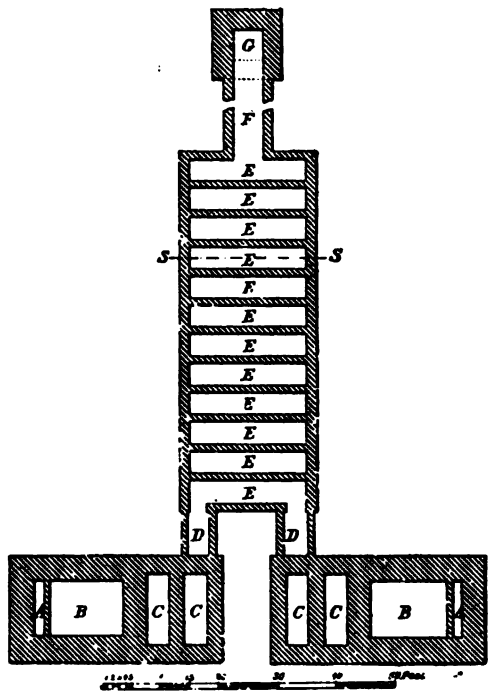


FIG. 121.—MODIFIED IDRIA FURNACE AT THE REDINGTON MINE.

years; it was made of firebrick. The condenser screen was built of red brick. The ore chamber B was 13 ft. by 10 ft., and 20 ft. high; its walls were 4 ft. thick. It had two openings H, Fig. 122, 32 in. by 20 in., for discharging the ore, 4 ft. above the ground on the outside. The car for discharging the furnace ran under a swinging apron of cast iron. The ore which had been treated was raked out over it. The holes in the fireplace and condenser wall are the starting-points of flues made in the charge. As the condenser side is constantly exposed to acid vapours it rapidly

wears away, and was renewed about twice in five years. To secure it against pressure, it was strengthened with an abutment. From the action of the ore and vapours the inside walls of the ore chamber were worn away to a considerable depth, making it larger than it formerly was. Beyond the ore chamber each furnace had two sandstone condensers C, 4 ft. wide, 13 ft. long, and 20 ft. high. Very little mercury ever condensed in them, not more than two flasks a week. They were connected with the main condensing chambers E in the upper part by a flue D supported on arches, which entered the top of the first chamber of the main condenser. The length of the flue D was 8 ft.

The furnaces are independent, so that each may work into the condenser without reference to the other, the connection being cut off at will The total length of each furnace, including fireplace, ore chamber, and the two condensing chambers, was

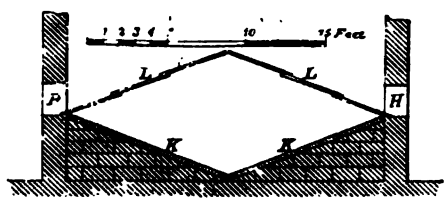


FIG. 122. SECTION AT S S OF FIG. 121.

36 ft. Its total width was 17 ft., and its outside height 24 ft. The main condenser E consists of twelve chambers, each of which is 4 ft. long, 20 ft. wide, and 20 ft. high on the outside, common to both furnaces. Each one of these chambers has an iron door H, shown in the section S S., Fig. 122 on each side at the bottom. The partition walls between each are 1 ft. thick; the outside walls are 18 in. They are braced by wooden beams 8 in. by 10 in. on the outside, which are tied together by $\frac{3}{8}$ in. round iron rods. These rods pass through the interior of the furnace, and must be carefully watched, as they are likely to become corroded through, and allow the wall to fall. All the walls of these chambers are very soon saturated with mercury. They rapidly become soft from the effects of the acid vapours; some of the side walls have been corroded to a depth of 6 in. only, while the partition walls as they are exposed upon both sides, have been corroded entirely through. Wherever they have been repaired with brick it has

been found to resist better than the stone. The relation of the condensing surfaces to that of the charging compartment is much too small. The ratio of the condensing volume to the volume of the furnace should be as twenty-four to one. In this furnace it was not half that amount. The condensing chambers are really only large enough for one furnace. On this account the walls have become super-saturated with mercury, and after a little exposure, even after they have been thoroughly cleaned, they still exude mercury in globules. The flue F, Fig. 121, leading to the chimney G, is built of cut stone, and is 250 ft. long. It is over 4 ft. in the largest diameter, with walls 2 ft. thick. It is built against the side of the hill. The chimney is also of stone, and is conical, 50 ft. high, 5 ft. in interior diameter at the base, and the walls 3 ft. thick.

The ore is charged by a windlass from above; the buckets used for charging are of iron 3 ft. in diameter and 30 in. high; the adobes are charged on platforms 3 ft. by 2½ ft., and 2½ ft. high, that carry 150 adobes weighing about a ton. To make the charge the ore compartment is first lined with adobes two to five thick piled together, but not crowded close. Inside of this the ore is placed in pieces from a size not larger than 12 cubic inches, down to one-quarter of this size in layers 18 in. to 20 in. in thickness. Adobe channels are then built, communicating with the openings in the wall of the fireplace and condenser side of the furnace. Their number depends upon the size of the ore, four for coarse and five for fine ore. They are made as flues to carry the heat into contact with all parts of the ore, and also have the effect of preventing the charge from packing together. A part of the soot that either has been worked, or is too poor to work in the retorts, is charged over every layer of ore. A new layer of ore is then put in over the adobe channels, and so on until the chamber is nearly full; over the top, fine ore mixed with poor soot is placed, and the whole is covered over with clay and ashes, which the heat bakes, so that it becomes hard after the furnace has been fired a short time.

When the charge is being worked these compartments are closed on the top with cast-iron pans, which are either filled with water or covered over with ashes, in order to prevent the heat of the

charge setting fire to the building above. The joints between the pans and the masonry are made tight with ashes to prevent the escape of the fumes. These pans are 5 in. deep, 4½ ft. wide, and 10 ft. long, or a little longer than the width of the furnace. They have two eyes on each end, so that they can be removed by a crane.

One furnace is charged Monday, and is fired Monday night, and kept burning until Thursday morning. Two cords of wood are burned during this time. The sulphur takes fire and answers for fuel for the most part, for there is always an excess of sulphur as iron pyrites in the ore. The other furnace is charged Tuesday and fired Tuesday evening, and kept burning until Friday. One furnace will thus be cooling on Saturday and Sunday, the other on Sunday and Monday. A small quantity of mercury runs in C on Tuesday, and in the first, second, and third compartments of the main condenser. It runs most freely Wednesday and Thursday. By Thursday a very small quantity of mercury is condensed in the extreme end of the condenser; 150 to 200 flasks are collected from both furnaces during the week, most of it from the fifth and sixth condensers. On Wednesday it commences in the fifth and sixth, and runs from this point until the furnace is discharged. Most of the mercury is obtained when the furnace is cooling down. When the draught is good, and the ore of moderate richness, the yield will not be less than 200 flasks per week.

The bottoms of the condensing chambers were originally built inclined towards the centre, as shown in section S S, Fig. 122, with the intention of having all the mercury in the condenser run out through a common opening made for the purpose at the bottom of the incline. The openings, however, became rapidly clogged, so that the flow of mercury was prevented, and instead of flowing to the outside of the furnace it became absorbed in the masonry. This method was abandoned; but as it was not worth while to change the construction of the furnace, iron pans were placed over this floor, sloping from the centre towards the outside, Fig. 122. These are placed 3 ft. above the old furnace bottom, so that there is a space under the bottom of the pans, which are made of three pieces overlapping each other. This construction is necessary, as when the walls get old and fall, as they frequently do, the

whole pan would be liable to be broken; but by this disposition only parts of it are. Tiles have been used to preserve the surfaces of the pans, which thus protected have lasted during four years constant use. Without this covering they last only two years, by which time they were entirely corroded by the acid fumes.

On Saturday the iron doors of the condensing chambers are taken off, and a workman collects the soot on the pan and from the walls about 2 ft. above its bottom, with a long hoe. What accumulates above is left until it falls by its own weight. Seventy-five to a hundred bushels of soot are collected in this way, which yields from five to twenty flasks of mercury. The soot is a mechanical mixture composed of ordinary soot, fine ore ashes, dirt, volatilised cinnabar, and sublimed mercury in very fine globules, carried over from the ore chamber. On account of the condensation of the steam from moisture in the fuel and ore, it is usually damp when removed from the furnace. The soot buckets are 33 in. high, 30 in. in diameter at the top, and 14 in. at the bottom. Each side of the condenser will fill such a bucket twice, a foot high.

It takes two men two days to clean all the condensers and work the soot. The amount collected varies very greatly. When the furnace is stopped for repairs, it cools; a large quantity of soot becomes detached as soon as the furnace is lighted again. The amount of mercury collected from the soot will vary from five to twenty flasks a week. The flue leading to the chimney is cleaned once in three weeks, and about a bushel of soot is taken from it.

It takes ten men one day to charge each furnace, and six men one day to discharge it. These men are paid $35 per month and board, which amounts to about $1.90 a day. There is one foreman, and one man to clean and repair the flasks, and fill them. They work during the day only. Each shift of twelve hours has its own foreman. When the draught is not good, a fire is placed at the foot of the chimney, which is 50 ft. high, to increase it. This fire is oftenest used when the ore furnace is allowed to get cool, but occasionally when the direction of the wind is unfavourable. Less than a cord of wood per month is required for its use. The draught depends for the most part on the way the furnace has been charged. In the fall, winter, and spring no

trace of mercury is found in the flue, but when the weather is hot it collects near the chimney, and in very hot summer weather it sometimes escapes. The accident of salivation is very rare except from the carelessness of the men. It generally occurs amongst those who are in the habit of using tobacco. Unless the men wash their hands constantly small quantities of mercury get under the nails and become attached to the hands, and in this way are carried into the pockets and boxes where the tobacco is kept. In Europe, men who eat in the works are not allowed free medical attendance, are fined, and otherwise punished; but this can only be done in the United States to a very limited extent. As soon as the workmen experience any sensitiveness about the mouth, they are instructed to go to the office, and are there furnished with a mouth wash consisting of two parts of cinchona, one part of tincture of myrrh, and three parts water.

Twice a year a general clean up is made. This is done by putting into the compartment C, Fig. 121, pipes 24 in. in diameter, and three in the flues leading to the chimneys. This is done both to cool and to ventilate the chambers while they are being cleaned. The walls of the condensers are scraped to remove the soot. As there is a considerable quantity of dust arising from the fallen soot, it is necessary for the workmen to protect themselves with a wet sponge, which is held over the mouth and nose by a thin cloth tied behind the head. No one man is allowed to work here more than fifteen minutes at a time. After having been thoroughly scraped, the walls are carefully washed. They are so thoroughly impregnated with mercury, that after having been scraped and washed every day for a month, when exposed to the sun for a few hours only they will show bright mercury globules all over the surface. As an illustration of the absorbing power of the masonry, the first time that the furnace was charged, the ore was picked. At the top of the charge 6 ft. to 7 ft. of rich soot was put in, but the heat did not reach it. The charge should have yielded 500 flasks of mercury; all but nine flasks were absorbed by the walls.

It takes eight men two days to wash the walls, flues, and condensers. The soot in the first condenser is often 6 in. thick, and in concretions which are sometimes hard and solid. This same concretion is sometimes found below the iron pans; it consists of

soot, hardened by the acid, and mercury in globules. In all the other condensers, however, it is soft and light. On the walls it is of very variable thickness, generally from 1 in. to 2 in., but sometimes is as much as 6 in. thick, when they have had a long run without cleaning. When the soot is very rich it may accumulate to 6 in. on the bottom of the pan. The richest soot is taken from the fourth, fifth, and sixth condensers. From a general clean up, as many as 100 flasks have been collected.

The furnaces were in use eight years, during which time they were repaired three times, parts of six partition walls having fallen down at the flue end. At the furnace end the wall is good. All the stone taken out from the condensing chambers during repairs is put on one side and treated as very rich ore. The furnaces originally cost $100,000 to $125,000. They are now abandoned at the Redington Mine. The masonry has been altered into Livermore's inclined continuous furnace for fine ore, the cost of the alteration being $1200. The average cost per week of reducing one ton of ore in the modified Idria furnace, No. 1 and No. 2, at the Redington Mine in 1874 was:

	$
5½ cords of wood at $5	27.50
36 days, furnacemen, fireman and assistant	86.00
20 days, labourers charging furnace	36.60
12 ,, ,, discharging furnace	23.15
8 ,, ,, working soot	14.55
8 ,, fireman	14.55
9000 adobes at $5 per m.	45.00
Total charge of 200 tons cost	247.35
Average cost per ton for reduction	1.25

In this expense it is calculated that there was not more than one-third of the charge made of adobes. If adobes only had been worked, the cost would have been $1.92.

New Almaden Works.

The reduction works or hacienda of New Almaden are situated on the Broad Gauge of the Southern Pacific and the Narrow Gauge, built by the Southern Pacific Coast Railway, and now owned by the former company. They are 62½ miles from San Francisco and 15 miles from San José, Santa Clara Co., California.

in the Santa Cruz Mountains. The railways terminate one, two, the other three miles from the hacienda. The mine is on Mine Hill, whose summit, at an elevation of 1700 ft. above the sea, is one mile west of the works in a straight line, and three miles by road, and 1300 ft. above them. The present workings are irregularly distributed over an area 5000 ft. north and south, and 6000 ft. east and west, and from the top of Mine Hill, and are 2300 ft. deep. From 1850 to 1884 they produced within 30 per cent. as much mercury as Almaden in Spain, and three times as much as Idria in Austria. During the year 1886-7, 19,550 flasks were produced from 38,194 tons of ore, showing the yield to be 1.96 per cent. at an actual cost of $27.32 per flask.

The mines were first worked for quicksilver in 1845, but the operations were on a small scale, and no record of them exists earlier than 1850. They have been, and now are, the most productive quicksilver mines in the world, excepting only the mine of Almaden in Spain. About five hundred men find steady employment in them, the work being actively prosecuted throughout the year. From the 1st January, 1864, to the 31st December, 1887, the number of feet of drifting and sinking in the mines of the company, as shown by the records, amounted to 224,922 ft., or 42.60 miles, at a cost of $1,918,457.29. This does not include the excavations made in extracting ore during this period, nor any of the expenses for doing it. The amount of ground opened up during the period from 1850 to 1864 was fifteen miles of drifting and sinking. These works are the oldest, the most enterprising, and the most successful of the quicksilver works of California.

The reduction works consist of eight furnaces, and include the most improved methods for working quicksilver ores, and may be considered as the most complete and perfect in every respect in the world. Most of the experiments which have resulted in the improvement of the processes have been originated and perfected here. They have been occasionally surpassed in production by other mines, but have always again come to the front. Both the mines and the works are admirably managed, and it is to the extended researches made on condensation at these works

FIG. 123.

f the Furnace Yard

of

LVER MINING COMPANY

EW ALMADEN

California.

B.C. denotes 'Brick Chamber.'
W.T. " 'Water Tank Chamber.'

that most of the real progress in the metallurgy of mercury in the United States is owing.

The mine is reached by a wagon road three miles in length, which is used to take up supplies. The ore is cinnabar associated with pyrites and some bituminous matter, and occasionally some native mercury. It comes down to the hacienda by a tramway with a self-acting incline. Its end is 60 ft. above the furnace, where the ore is discharged into ten bins, shown on the plan of the works, Fig. 123, built on the side of the hill at an angle of about 40 deg. When the ore comes from the incline it is dumped into the bins at back of the furnaces by turning the wagon half way round, attaching a chain to the bottom and raising it with long levers, which are permanently placed on the top of the bin. Each bin is arranged with screens to size the ore. The bottom of the bin is on a level with the top of the furnaces, and is connected with them by railways.

The ore from the mine, just as it is taken in the labores or stopes, is brought in the mine cars to a long and narrow shed called a planilla. The waste rock arrives at the same level, but is switched to the dump heaps to the left. The ore car continues on a trestle nearly on the level of the peak of the roof to the dumping places. The planilla is 280 ft. long, 63 ft. wide at the further end, and a little narrower at the mine end. In the centre of this is a trestle work 16 ft. high, with bents 32 ft. apart. Between each bent an iron screen is placed at an angle of about 45 deg. The screen is 48 in. wide, with a false side made of 2-in. plank 12 in. high. The screen bars are slightly chamfered with their thin edge below, so as to insure a complete separation of the ore. Every 24 in. they are secured by an iron bar which is bolted to a 3-in. square joist on the under side. At the head of the screen are three rails spread, so as to throw the ore over the whole surface of the grating. There are nine of these screens in the building. On each side of the screen is a passage to sort the ore, but on one side there is a space 20 ft. wide with posts 7 ft. between centres with openings in the roof, in which the larger pieces of ore are placed after they are picked, and are broken to about 9 in. square. This ore is called granza. On the other side of the planilla and on each side of the bents

the ore is broken and sorted. The space between each bent is numbered, so as to show to whom the pile belongs.

All the ore which passes over the screens is broken to about 9 in. square, throwing away the poor rock. This ore is called granza, and usually runs from 6 to 8 per cent. What passes the screens is called tierras; these belong to the company. The granza are paid for to the miners. Besides the mine supply, the old dump heaps are being worked over on the planilla on similar screens, but in the dry season only. The large pieces which pass over the screen are washed in a tank with ore forks 12 in. wide, having tines 1 in. apart. All the pieces which show the least sign of cinnabar are put on one side and called terrero; these are about 6 in. in size. The fines which pass the screen are called dump tierras. The granza and tierras ores are weighed as they leave the planilla. The other ores are estimated by volume. They weigh about 85 lb. to the cubic foot, or 23.5 cubic feet to the ton. All of this ore goes from the planilla to the hacienda, nearly two-thirds of the ore being fine. This is done over an automatic incline, at the end of which the ores are stored in bins, shown in the plan, Fig. 123, according to sizes. There are ten of these bins, one large one, which is 50 ft. long with vertical sides; two 15 ft. wide, with logs on the bottom to delay the descent, inclined at 45 deg.; one with six screens, 2 ft. long and 2 ft. apart, set at 45 deg.; two 8 ft. wide with cast-iron screens, with meshes $1\frac{1}{2}$ in. square, set on the diagonal. The screens are 2 ft. long, with five rows of screens on the side, and six with two rows in the middle. On the top there are four screens: two granza with log bottoms, 15 ft. wide; one tierra, 6 ft. wide, with cast-iron holes $1\frac{1}{2}$ in. square, partly set square and partly on the diagonal; one tierra, about 6 ft. wide, with suspended impediments to prevent the ore passing down too quickly, and boards to direct the passage of the ore. One of the impediments is a board, the other is a vertical screen, hung so that the bottom was smooth. Experiments with these screens have shown that the grizzly pattern is the best, as a piece of ore may be small enough at one end to lodge in the square, but too large to pass, and thus obstruct the square hole, while the same piece of ore would pass through the grizzly. The granzita and

tierra ore bins are one above the other, and are reached at different levels. The granza are simply large bins. These are so arranged that they are reached from any level, and go to any furnace. At the foot of the incline there was formerly a large ore bin, but it had to be taken down. At the hacienda a further sorting takes place on the ore bin screens, making an intermediate variety of ore, called granzita. This is carried further by covering the wagons which supply the granzita and tierras furnaces with permanent screens, so that no ore of a larger size can be charged into them. The Table below shows the different varieties of these ores:

At the Planilla.

	Quality.	Size of Ore in inches.	Per cent. of Mercury.	Determination.
Granza . .	rich	9 by 9 by 3 to 4	6 to 8	weighed
Tirrero . .	poor	6 by 6 by 3 to 4	1 to 3	volume
Tierras . .	poor	1 by 1 to dust	1 to 3	weighed
Dump tierras .	poor	1 by 1 to dust	1 to 3	volume
At the Hacienda.				
Granza . .	rich	9 by 9 by 3 to 4	6 to 8	weighed
Granzita . .	poor	3 by 3 by 1	1 to 3	volume
Tierras . .	poor	1 by 1 to dust	1 to 3	volume
Dump tierras .	poor	1 by 1 to dust	1 to 3	volume

The finer varieties of ores are estimated by volume, as the quantity of moisture they contain is so very variable. They are so poor in quicksilver, and the difference in weight is so small between the raw and the spent ore, that it would lead to error if they were weighed without being carefully dried, which is impracticable. As the capacity of the ore bins is only sufficient for the current work of the furnaces they are supplemented with ore sheds, in which the winter's supply of granzita and tierras ores are stored during the summer for the winter's work, as, if left exposed, they would absorb too much moisture. Without this precaution they would have to be dried, as was formerly done on the top of the old non-continuous furnaces, but is not now practicable. The storage capacity of the works is about 100 tons of ore, and 500 to 800 tons of granzita and tierras on the planilla. And at the hacienda 5000 tons of tierras, 800 tons of granza

and 1000 tons of granzita, or about 3000 tons of ores of all grades.

All of the furnaces are under roof in buildings with open sides. Nos. 6, 8, 3, 7, and 9, Fig. 123, on the west side of Alamilos Creek, and Nos. 1 and 2 on the east. An elevator between 3 and 8 serves to raise the ores that have been stored or sun-dried, or that have been dried in the driers of No. 6 or No. 3 furnace, to the level of the top of the furnaces. The creek is crossed by two bridges: one a high one, from the ore bins to half the heighth of Nos. 1 and 2, and the others at the level of their floors. Nos. 1 and 2 have their own buildings, and have an elevator also, and are entirely independent of the furnaces on the opposite side.

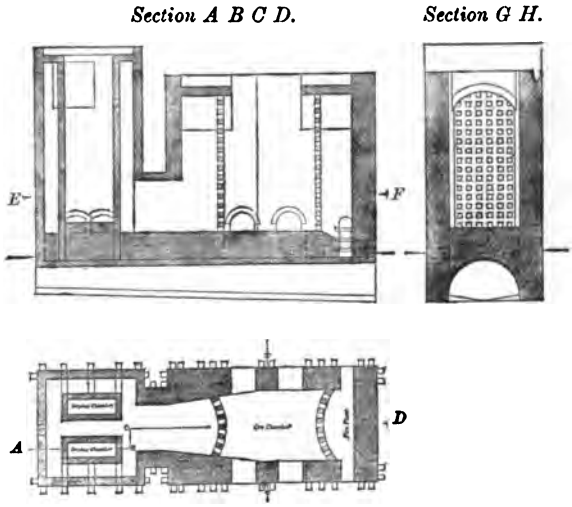


FIG. 124. INTERMITTENT FURNACES AT NEW ALMADEN.

Formerly all the furnaces of the hacienda were of the modified Idria type. They were numbered as they were built in serial order. There were formerly six of them. In 1873 one of them was torn down to make room for the experimental Exeli furnace, so that there remained only five, four of a capacity of 50 tons per week, and one of 100. In the summer of 1888 there was only one left, which was furnace No. 6. There are now four different kinds of furnaces adapted to the treatment of the different grades of ore, their positions having been determined by the growth of the works. As most of them have been built on the sites of old

intermittent furnaces, the new have retained the numbers of the old furnaces, though the numbers are not always consecutive. The plan of the works as it was during the summer of 1888, is shown in Fig. 123. Fig. 124 shows the furnace known as No. 6, which is the only remaining intermittent furnace at New Almaden. The construction of the furnace is essentially the same as at the Redington Mine, except that drying chambers for the fine ores have been introduced to expel the moisture from them by means of the waste heat. This chamber has two discharge doors on each side. The ore chamber is 12 ft. long, 9 ft. wide, and 18 ft. high, and holds from 90 to 100 tons. The ore is charged in baskets let down from the top. Channels are made with the coarse ore to connect the fire and flue screens. This was formerly done with adobes, which are now no longer made. These channels are covered with ore to the depth of 2 ft. to 3 ft., another series is made above, and so on. To make an even distribution of the heat, which naturally rises in the furnaces, these channels are made larger and nearer together at the bottom than at the top and the amount of small ore is increased there. Near the top fine ore and soot is used for the same purpose. The discharge flue into the drying chamber and condensation apparatus is made only 7 ft. high instead of the whole height of the furnace. The filling of the furnace takes eight men one day. When full, the discharge doors on both sides are fastened up and luted. The top is covered over with old sheet-iron, over which straw is laid to prevent anything falling into the ore. Over this clay is packed to the depth of 2 in. or 3 in. This cover and the lute on the discharge doors require to be carefully watched during the firing, and any cracks which show must at once be made tight. The furnace is fired for four days and a half, and then allowed to cool for three and a half days, eight days altogether. During the cooling the doors of the fireplace are opened to allow the air to freely enter. At the end of eight days the ore will be cool enough to be handled. The top is then removed, and the discharge doors opened. The spent ore, or "slag," as it is called, is drawn into iron cars. The discharging of the furnace takes four men one day, so that each furnace can be turned only three times a month. It was formerly done in a shorter time, but the yield

was less, and the cases of salivation frequent, so that the increased production of the furnace was dearly purchased.

When the channels were made of adobes, the conduct of the furnace was slightly different. The adobes were 10 in. by 4 in. by 5 in. in size, and were made of the fine ore mixed with clay. They were made by hand in wooden frames holding six at a time. The moulds were $4\frac{1}{4}$ in. by $5\frac{1}{4}$ in. by $10\frac{1}{4}$ in. inside measure; and the sun-dried adobe weighed about 12 lb. They were lowered in Mexican ore baskets by hand from the top of the furnace, and were placed over the floor of the ore chamber, one adobe thick, and the entire height of the sides; 2 ft. 6 in. to 3 ft. of ore was then charged, then a layer of adobes, in which six flues were placed from one side of the ore chamber to the other, corresponding with the pigeon-holes in the arched walls on both ends of the ore chamber; these were covered with ore, and so on to the top. These were covered with a layer of adobes, then 2 in. or 3 in. of soot, then ordinary clay and mud to fill the furnace even with the top, and over this ashes. When during the charge any cracks occurred, they were carefully filled. When the furnace is filled in this way one-third of the charge will be adobes.

As adobes had to be made here the year through, and must be dried in winter, the brick condensers were covered on the top with iron plates for drying them. Occasionally the furnaces ran on adobes alone. In 1875, the cost of making the adobes was 50 cents per ton, and that of handling them afterwards, drying, &c., 45 cents per ton. They cost about $5.70 per thousand. The last of the adobes were treated in this furnace in October, 1882. Since then they have ceased to be made. It takes six men one day to charge the furnace, and four men one day to discharge it. The charge is made by the company, as it must be done with great care, but the discharge, as it requires no supervision, is done by contract. It takes one man part of twelve hours to keep up the fire, which is kept up for ninety-six hours, and during this time 18 cords of wood are burned for 100 tons of ore. The wood costs $4 to $6 per cord. The ore is discharged twelve hours after the fire is drawn; twenty-four hours afterward the soot is removed.

The furnace now works on granza ores when convenient. In 1886 it worked forty days, and produced 126,301.2 lb., or 1673

flasks of mercury from ores yielding 6.815 per cent. In 1887 it worked thirty-nine days, and produced 116,815.5 lb., or 1527 flasks of mercury from ores yielding 7.3 per cent. In 1888 it worked twenty-three days, and produced 100,368 lb., or 1312 flasks of mercury from ores yielding 9.558 per cent.

The drying chambers, Fig. 124, for the ore are simple shafts. The ore is charged and heated by the vapours, which are cooled. The heat penetrating through the bricks brings the ore up to such a temperature, that the moisture is partly evaporated. When well heated through, it is spread out in front of the chambers, so that it becomes thoroughly dried. The system of condensation in brick chambers, Fig. 134. is still preserved in this furnace. There are eighteen of them, which are numbered consecutively, 3, 6, 9, and 13, are now cut out, and the interval arched so as to secure the stability of the structure; as they are 4 ft. wide this leaves 4 ft. between the condensers. Nos. 1 and 2 are water backed. After No. 18 there is a brick tower and then a wooden flue, which connects with a brick one leading to the Guibal fan. All the connections between the different condensing chambers of this furnace are earthen pipes 18 in. in diameter. Between Nos. 2 and 4 there are seven pipes; between Nos. 5 and 7, five; between Nos. 12 and 14, seven. At the top and bottom of each one of these chambers there are openings closed in the hot condensers by iron plates, and in the cooler ones by glass window frames, which are 30 in. by 22 in. As the furnace was not running in July, 1888, it was used as an air drier for tierras ores.

With ordinary ores, when the furnace was running the mercury commenced to condense fourteen to sixteen hours after the furnace was lighted. If the ore was exceptionally rich it sometimes condensed in eight hours. Before the chambers were cut out it appeared first in chambers No. 1; in Nos. 1, 2, 3, 4, and 5 it ran for ten hours; after this it ran in Nos. 3, 4, 5, 6, and 7. It appeared in No. 7 on the second day, after which it stopped running in the first four condensers, but continued in all the others. The greater part collects in Nos. 10 to 18. Before the fire is drawn, one-half the mercury is condensed, it then gradually diminishes until it ceases; a small quantity of mercury still runs from the pipes, the flow being kept up by the dripping

from the soot on the walls. The mercury condensing runs into an outside trough, from which it is discharged into a reception basin. The brick condensers were formerly closed with iron doors. These have been entirely abandoned except for the first four or five chambers, which are too hot for wooden ones, and are now replaced by ordinary house window sashes, with six panes of glass. After their adoption there was a very decided increase in the percentage of mercury condensed. Since the adoption of the interval between the condensers by cutting out the chambers and the water backing them, the mercury commences to condense in the water-backed chambers Nos. 1 and 2.

Very large condensers made of wood and glass were patented in May, 1874. They were very much cheaper, and it would seem a much better condenser than those formerly used. They require no water, and are applicable everywhere; for the want of water in many places in California is a serious drawback to working. Four such wooden condensers were at work at New Almaden in 1874. Each one had a volume of 2640 and 1196 square feet of condensing surface, or altogether 10,560 cubic feet of volume and 4784 square feet of surface. They cost $1500, which is the smallest cost for any such amount of condensing surface known. They have since been abandoned on that side of the creek. It was afterwards found necessary to cut them up into chambers of about twice the size of the brick ones, but they were subsequently abandoned on these furnaces altogether.

The last brick condenser of the present system is a very small one, and connects by a wooden flue 66 ft. long, with a brick tower, then by another wooden flue, 36 ft. long, with another brick tower, and then by a brick flue, 41 ft. long, with the flue leading to the Guibal fan.

The following Table gives the working of both sizes of these furnaces on ore for the year 1875:

		50-Ton Furnace.		100-Ton Furnace.
Tons of ore treated	granza	19.0	...	890.5
	terrero and granzita,	142.5	...	1149.5
	adobes	1485.5	...	1845.5
		1647.0	...	3385.5
Percentage of mercury contained		1.43	...	2.76
Number of flasks produced ...		618	...	280.6

The cost of labour and materials was:

Furnace hands	... $1.25 to 2.50, ordinary labour, per 12 hours shift.
English coke	... 15.00 per ton, the ton is of 2000 lb.
Charcoal ...	... 12.00 per ton, the ton is of 2000 lb.
Bituminous coal	... 8.94 per ton, the ton is of 2000 lb.
Pine or oak wood	... 5.75 to $6.00 per cord.

The expense of treatment was:

	50-Ton Furnace.	100-Ton Furnace.
	$	$
Labour	2308.80 ...	3252.50
Wood	3369.60 ...	4933.20
Coke and charcoal	225.60 ...	407.00
	5904.00 ...	8592.70
Per ton of ore	3.58 ...	2.21
Per flask of mercury	9.55 ...	3.06

The cost of the making and treatment of one ton of adobes for the year 1875 was:

	$
Making adobes	0.52
Transportation and storage of adobes	0.47
Wood	2.01
Labour...	1.04
Repairs of the furnace	0.23
	4.27

The cost of mining was $1.59, and of transporting to the hacienda $0.38, or $6.24 for mining and treating the ore. In 1879 the cost of treatment of adobes was $2.33, nearly one-half less. The cost of treatment of one ton of 7.2 per cent. ore in this furnace without adobes in 1888 was $1.38, of which two-thirds was for fuel, and one-third for labour, but the furnace worked only twenty-three days.

In the year 1874 there were six of the brick non-continuous modified Idria furnaces in operation, which produced in the year 1873, 11,042 flasks of mercury, of 76½ lb. each; in 1874, 9084, in 1875, 13,648, and in 1876, 20,549. Four of these furnaces had a capacity of 50 tons, one of 70 tons, and one of 100 tons a week. They ran a week, each furnace making four charges a month. Each of these furnaces had its own condensing apparatus comprising 16 to 22 brick condensing chambers, the numbers varying with its ore capacity. The bottom of the chambers inclined from the centre towards the outside, and the

floor was built of brick covered with cement. Besides these brick condensers, there were four or five wooden ones attached to each furnace, which were 26 ft. long, 14 ft. wide, and 20 ft. high, divided into six compartments of equal size. As the result of the addition of these wooden chambers, one of the flues which formerly yielded from 60 to 100 flasks in the general clean up, yielded in 1873 only two flasks, the largest part of which was found near the outlet from the condensers. The more modern system is a great improvement even on this.

The soot was collected in the same way as at the Redington Mine. There, from four charges, eighteen flasks of mercury were obtained. The amount produced directly from the ore was 474 flasks. The soot collects in all the condensers from No. 4, on, but the most of it collects about the middle. About a barrelful per week is collected from each condenser; one man does the whole cleaning up. The soot was formerly worked, as in other places, on an inclined plane with a wooden hoe. A certain quantity of mechanically mixed globules were separated and ran into a vessel containing a little water which was prepared to receive it. The soot remaining behind was treated with lime. This method, which was the one formerly in general use, has been abandoned here, as it consumes a great amount of time and labour. The soot was, until 1887, charged into a hemispherical boiler 3 ft. in diameter, and was mixed with boiling water and wood ashes, about half the bulk of the soot being added in ashes. The whole is thoroughly stirred for three-quarters of an hour. It is then allowed to settle, and during this time another boiler is worked. The mercury collects in the bottom of the vessel and flows as it collects from a syphon pipe, so that it is entirely free from any mechanically mixed material. The water is drawn off afterwards from the boiler, the soot remaining behind is dried and placed on the top of the furnace where the flame passes, as it still contains some mercury. This process was patented October 4, 1873. In all essential points it is similar to the Manhattan method, which seems to have the advantage of having dry soot, and of requiring a little less labour. It takes one man, working ten hours, four to five days to work the soot of six furnaces. The fuel which is used is

wood, or ordinary charcoal made from oak, or a mixture of coke and charcoal; 75 lb. of coke and charcoal mixed are required to treat 3000 lb. of ore and adobes. In 1875 the net cost per flask was $19.52.

The furnace which is used here and at the Redington Mine is a very great improvement on the old Idria furnaces, as the ore chambers contain no arches, which are constantly liable to break down. The method of charging the large and small ores and making air channels of the adobes or large pieces of ore, prevents the charge from packing, and gives all the space required for the circulation of the air and heat, and permits the use of poor ores. The cost of making the adobes is $5.70 a thousand, $5 of which is for labour. It was formerly considered that these adobes were essential to the working of the furnace, and that the air channels could not be made of large pieces of ore, and if they were there would be no certainty that they would last, but furnace No. 6 worked five years with these channels made of large pieces of ore without experiencing any inconvenience. Neither the large pieces of ore or the adobes change their form during the treatment, and the air channels once made remain until the charge in the furnace is disturbed either by accident or to discharge it. The making of the adobes, however, is an expense, which in the furnace of the future will be entirely avoided. There is little doubt that this style of furnace will disappear altogether, and will only be known in the future as one of the curiosities of metallurgical history.

The following Table gives the furnace expenses of a flask of mercury at New Almaden in 1876 in furnaces Nos. 2, 5, and 8. No. 2 treated adobes alone; Nos. 5 and 8 treated the ore as it came from the mine without adobes.

Furnace No. 2.		$
Amount of wood per flask, 1.03 cords at $6		6.18
Amount of labour per flask		3.77
2,562,000 lb. of tierras represent 213,500 adobes with cost; original cost of making	$ 3.00	
Breaking	0.60	
Carting and storing	2.10	
Cost per thousand	5.70	
213,500 adobes at 5.70 per thousand, cost $1216.95; therefore the cost of tierras per flask of quicksilver		3.80
Total cost of one flask produced in No. 2 furnace during eight months' operation		$13.75

Furnace No. 5.	$
Amount of wood per flask, 0.65 cords at $6	3.90
,, labour per flask	4.02
Total cost of a flask of quicksilver produced in No. 5 furnace during ten months' operation	$7.92

Furnace No. 8.	
Amount of wood per flask, 0.47 cords at $6	2.82
,, labour per flask	1.69
Total cost of a flask of quicksilver produced in No. 8 furnace	$4.51

The great difference in cost between furnaces No. 2, No. 5 and No. 8, is owing to the use of adobes in No. 2. The difference between Nos. 5 and 8 is owing to the improved method of working adopted in No. 8.

CONTINUOUS FURNACES.

The inconveniences of running intermittent furnaces were so great, the loss in mercury so large, and the fact that of the ore produced by the mine nearly two-thirds was fine, and had to be made into adobes before it could be treated, and that this added nearly 70 per cent. to the cost of working, led to the desire for improvement. When the price of quicksilver rose, and the discovery of the great mines of North California had been made, the greatest activity was caused in the invention of new methods of extraction in which not only all the ore could be treated without the necessity of making adobes, but in which the work was to be continuous. The same necessity which was so pressing in California existed also in Europe, and led there to the invention of a number of types of furnaces, most of which were never built in the United States. In the year 1871 Bergrath A. Exeli adapted Count Mumford's lime kiln to the treatment of cinnabar, which was so successful at Idria in Austria that Mr. Randol, at new Almaden, in 1874, constructed one upon the spot indicated as No. 7, which worked so well that the following year No. 9 was built. These furnaces were built in a mantle of sheet-iron, and were called "monitors" or "iron-clads," in allusion to the war vessels known under that name. The introduction of these continuous furnaces for large ore was quickly followed by the invention elsewhere of the Luckhart furnace, copied after a Swedish iron ore calcining furnace, and the

Knox furnace, which at one time seemed likely to displace every other, but which, partly from the extravagant claims of its owners, and partly from the sudden fall in the price of mercury followed by the great decline in the production of the mines, seems, like many others, to have disappeared. Both of these last furnaces proposed to treat the coarse and the fines together, but failed to do so. This was followed by the invention of the Livermore furnace at the Redington Mine in 1874, and of the Hüttner-Scott granzita and tierras furnaces at New Almaden in 1876. The main ideas of both the Livermore and Hüttner-Scott furnaces was taken from the Hassenclever furnaces, the Hüttner-Scott from the earlier shaft furnace type, and the Livermore from the inclined one so much used in the roasting zinc blende for the manufacture of sulphuric acid. This last furnace was for some time very successfully used at the Redington Mine. In 1878 a furnace was started at New Almaden, known as No. 3, which was of the same type as No. 5, but much larger, and destined to treat the medium-sized poor ores or granzita, but it was found to be better suited to the tierras or fine ores, and is now used for that purpose. In 1879, No. 2 for treating the granzita was built, and in 1880 a double one, No. 1, was built near it. These six furnaces at New Almaden, Nos. 1, 2, 3, 7, 8, and 9, are the present plant of continuous furnaces at New Almaden, and represent the latest improvements in treating ores of mercury of all grades in California. While constant improvements, either in furnace construction or in condensation have been made at New Almaden, no progress has been made elsewhere for a number of years, owing mostly to the fact that since the great fall in the price of mercury most of the mines in California, as shown by the Table opposite page 901, have closed down. New Almaden at the present time produces three-fifths of the entire mercury production of the United States. The description of the treatment for all the mines, except New Almaden, dates some years back, and in most cases has quite as much historical as metallurgical interest.

With the single exception of the Livermore furnace, all the continuous furnaces are vertical shaft furnaces. They do not differ essentially in principle, though they do differ very widely in detail,

which is sometimes a difference of construction, and sometimes of condensation.

A great variety of these shaft furnaces have been patented in California. They are referable, however, to five general types, the Luckhart, the Knox, the Exeli, the Hüttner and Scott, the Granza, the Granzita, and the Tierras furnaces, and the Livermore inclined fine ore furnace. A former legal decision pronounced the larger part of the vertical shaft furnaces to be an infringement of the Knox patent, but this decision has since been reversed.

The Luckhart Furnace.

The Luckhart furnace is similar in all respects to one of the furnaces formerly used for calcining ores of iron. It consists of a shaft with a roof-shaped firebox of cast iron, about 4 ft. above the bottom of the shaft, with openings on the two sides only, which go through the furnace from side to side. The slant of the covering of the firebox inclines from the centre towards the side of the furnace, so that the ore as it passes down is divided at this point. The discharging door is sufficiently far below the grate to insure that the ore will remain long enough in the furnace, both to distil its mercurial contents, and to be sufficiently cooled afterwards, to be discharged with safety from the bottom of the furnace into an open wagon. By this system, which is common to all the continuous furnaces, all inconveniences arising from the discharge of hot ore not entirely worked are avoided. It is provided at different heights with peep-holes to watch its operation. The furnace used at the Sonoma Mine was capable of treating from 10 to 15 tons of adobes a day. It could have been used on ores alone. The richest ore ever treated at this place did not contain over 1 per cent. The serpentine in which the mercury occurs at that mine crops out near the surface, and is very much decomposed. Where it is exposed to the weather its débris have been washed away, so that soil for some distance from the outcrop is more or less mixed with free mercury and a very small quantity of undecomposed cinnabar. Where the rock has been mined so as to reach the unaltered parts of it, more or less free mercury is found mixed with the cinnabar.

Nine-tenths of all the material treated, however, consisted of the dirt, which was made up into adobes. These contain from ¼ to ½ per cent. of mercury. The rock itself does not contain more than 1½ per cent. The adobes are made by Chinamen, who mix the earth with water to a thick paste, and pack it into wooden moulds 5 in. wide, 4 in. long, and 4 in. high. A frame containing eight of these moulds, placed on a board a little longer than the exterior of the frame, has the mud thrown upon it, which is then patted with a shovel; the dirt is then scraped even with the top of the frame, the frame is drawn up, and the adobes left on the board as they were; this is then removed to a convenient place, with the adobes upon it, to dry in the sun, which takes about two days. One Chinaman can make about 1000 a day. It takes two Chinamen at $1.25 per day to pick as much dirt as six men can use for making adobes. When ready for use they weigh about 8 lb., and are a little larger than an ordinary brick.

The furnace is charged with a circular hopper 22 in. in diameter, and 26 in. deep. Five or six of these hoppers full make a charge. The hoppers are covered over with a hydraulic cover for safety, but even when this is removed the draught is so strong towards the interior of the furnace, that it will easily draw a piece of paper into it. One and a half cords of oak wood a day is all the fuel used. The wood is worth $3 a cord. The spent ore is discharged from the bottom of the furnace quite black, but the last of it is occasionally dull red. It is drawn every one and a quarter hours into iron wagons, which hold about ½ ton. There were seven men engaged in making adobes, one Chinaman and one engineer to each shift of twelve hours at the furnace. The Chinaman is paid $1.50 per day and finds himself. The engineer is paid $65 a month and board. They help each other in charging the furnace and the fireplace, and in drawing and dumping the ore. Besides this one man is engaged chopping wood.

The condensing apparatus consisted of three sheet-iron cylinders, 14 ft. high and 6 ft. in diameter, which were placed on the ground 5 ft. apart. Around the top of each of the condensing cylinders, there was a lead pipe 1 in. in diameter pierced with small holes 3 in. apart. When there was a plentiful supply

of water it was allowed to flow continuously, and discharge over the outside of each cylinder; when there was a lack of water, it flowed only over the last condenser.

The vapours from the furnace passed into the top of the condenser from a pipe 20 ft. long, and out at the bottom. In the interior of the connecting pipe there was a partition about 2 ft. long on which a considerable quantity of soot collected. The bottoms of the cylinders inclined towards the centre, from which a pipe lead to an iron kettle on the outside which was filled with water. The soot collected on the sides, and the little which fel to the bottom was carried out by the condensed moisture, whicl was but slightly acid, as there is but little sulphur in the ore.

A blower 3 ft. in diameter is placed beyond the last condenser, which makes from 100 to 120 revolutions per minute. In case ore not made into adobes is charged, the blower must run faster. The engine is 12 horse-power, which is twice the power required for a single furnace. Beyond the blower there is a wooden box 6 ft. by 12 ft. by 8 ft., into which the blower discharges. This box is divided in the middle by a partition which goes to within 2 ft. of the bottom, in order to break the current. From this box a flue 2 ft. square and 100 ft. long, ends in a wooden chimney 20 ft. high. Nearly one flask of mercury a day is collected from the first two iron condensers, which is about equally divided between the two. Generally a flask a month is collected from the third. About once in three weeks the furnace is stopped, and the manhole of the cylinder is opened to collect the soot, which will yield about ten flasks of mercury that is mechanically caught in it. The rest is charged in an iron pan, 9 ft. by 3 ft., with a gutter in the centre. The pan is set at an angle, and heated to 120 deg. Fahr., and the soot is then mixed with lime or ashes, and stirred, the mercury flowing into a kettle at the bottom of the incline. About 70 per cent. of the mercury in the soot is collected in this way. The acid water flowing from the condensers carries off some floured mercury and soot, this is collected in a wooden tank 12 ft. long, 6 ft. wide, and 4 ft. deep; the settlings are treated as soot. A little soot is taken from the blower. The further it is from the furnace the poorer it gets. The richest contains 60

per cent. of mercury, and the poorest about 3 per cent. As the furnace is very small, and the product not large, the condensation appears to be very good. There is so very little sulphur in the ore that the sheet iron is very slightly acted upon. With this system of employing large iron cylinders cooled with water with a converging bottom, there seems to be very little opportunity for the action of acid. A few large compartments, however, cannot give as good results as a large number of small ones, since a perfect condensation requires maximum surface, and does not depend so directly upon volume.

This furnace, for one cause or another, does not appear to have been very successful. In a great many places where it had been erected it was either torn down or modified into another type.

The Knox Furnace.

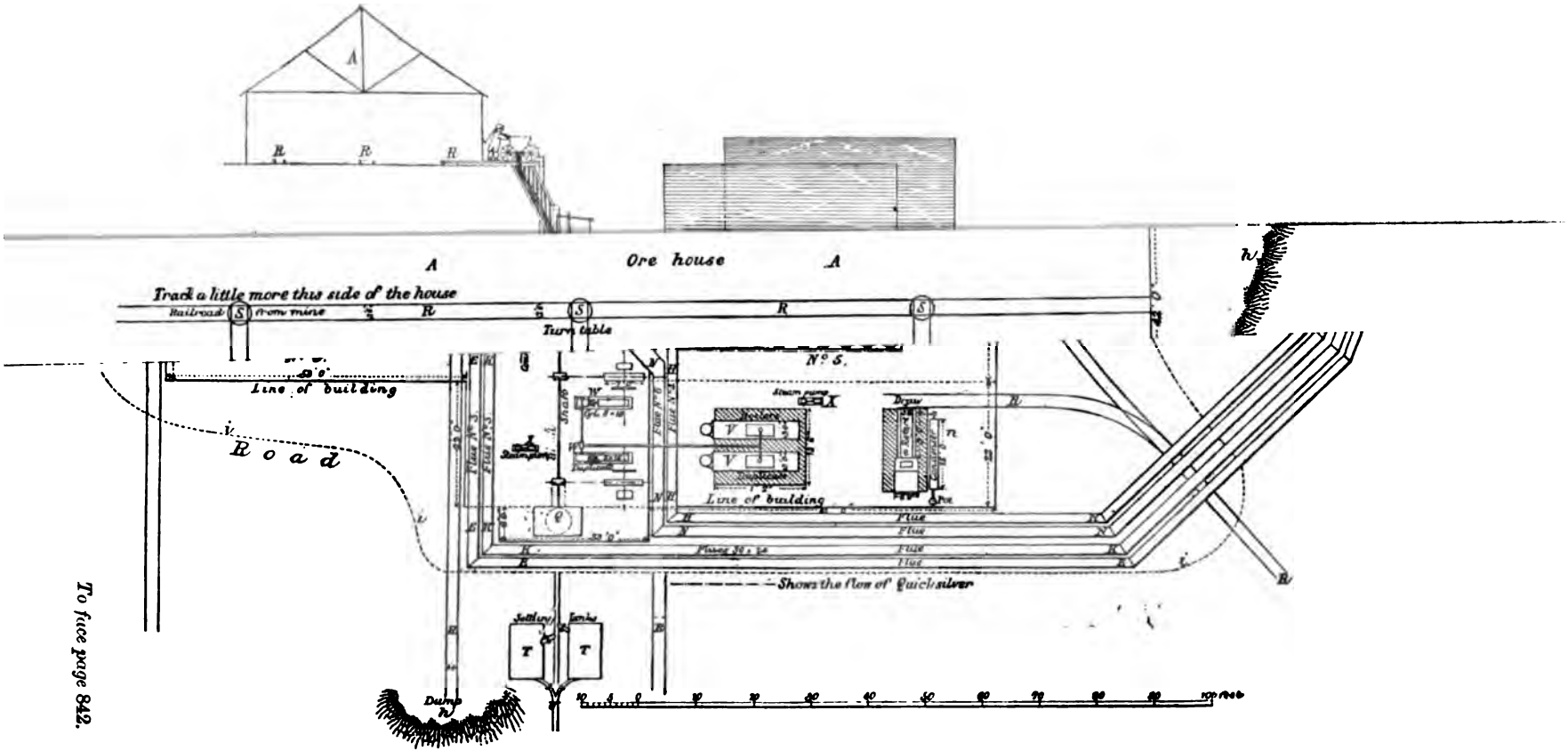
At the Redington Mine the ore occurs between a hanging wall of sandstone and a foot wall of serpentine. The cinnabar is found mixed both with chalcedony and ordinary quartz. Heavy seams of clay extend in front of, over, or running through the deposit, which in some places is several hundred feet thick. At this mine, there were four Knox furnaces, each treating 24 tons in twenty-four hours; and Livermore's furnaces, of the same capacity, two of which were made out of the modified Idria furnaces. The amount of ore treated per week was from 1000 to 1200 tons.

Previous to the introduction of the Knox furnace, while the modified Idria furnaces were the only ones in use, all the ore dressed was up to 5 per cent. Afterwards, ore yielding not more than 1½ per cent. as it came from the mines, was treated. No adobes are now made at Knoxville. The Knox furnace, Fig. 125, is a shaft furnace with a fireplace on the side. The height of the furnace is 39 ft.; the bottom of the fireplace is 17½ ft. from the bottom of the furnace, so that the fireplace is about the middle of the shaft. The furnace is rectangular in shape, the top is 2 ft. square, and continues of this size for 2 ft., and then widens out on one side until, at the depth of 4 ft., it attains the width of 7 ft.; while in the other direction it continues straight to a depth of 10 ft., and then increases gradually until it becomes 7 ft. wide

at the fireplace. At this point there are two chambers arranged in the masonry, one on one side for the fireplace, and one on the other *l* for the introduction of the pipe *j* carrying off the fumes. Here commences a series of six retreating arches which support the masonry of the furnace, four of which retreat 2½ ft. on either side. The two others, which are above, are much smaller, and reach to within 3 ft. of the top of the furnace. In the furnaces, as originally constructed, there were only three arches, and in some cases their number was reduced to two. In those afterwards built, there were four of them with six openings on each side. It has been found that a depth of 7 ft. in the shaft is too great a distance across the furnace for the heat to traverse, so that in the later constructions this distance has been reduced to 2 ft., so that the section of the furnace at the fireplace is a parallelogram 7 ft. by 2 ft. With this construction a much larger quantity of fine ore can be treated, and this form is now adopted. By the enlargement of the furnace in the centre, the ore, as it comes to the point where it is to receive the highest temperature, is made to spread out suddenly, so that each particle of it comes in contact with the heat.

The shaft continues to the bottom of the fireplace side, the same size as on the other side; it contracts equally upon both sides, so that at the bottom, where the ore discharges, the width is only 2 ft. The fireplace *e* follows the retreating arches, with a width of 36 in. to the top of the second arch, and then diminishes to 30 in. at the top of the third, and continues so to within 3½ ft. of the top of the furnace. Between each of these arches, on both sides of the furnace, there are openings 6 in. wide, the whole width of the arch. The aspiration of the blower draws the flame from the fireplace to the vapour chamber *l* on the opposite side, the heat being always sufficient to keep the ore at a dull red. This construction of the fireplace is unnecessarily complicated. The same effect might be produced by having the fireplace entirely outside of the furnace. The fireplace is so arranged that it receives air as it enters the bottom of the furnace, through a series of channels *f*, so that it is heated by the masonry, and a small saving of fuel is effected in this way.

There is a disadvantage about the construction of the fireplace,



125. KNOX FURNACES AT THE REDINGTON QUICKSILVER WORKS, CALIFORNIA.

To face page 842.

which is that in order to keep the back of the charge at the red heat which is required, to be certain of perfect reduction, the front and the middle must be raised much higher, and while there is little or no danger that the ore will ever be fused, there will be a greater amount of fuel used than there is any need of, and a greater deterioration of the furnace in the hottest parts than is quite necessary.

Furnaces with double fireplace and ore discharge, with a mercury discharge above the fireplace on one side, have been built, the dimensions of the shaft being 10 ft. by 5 ft. from fireplace to fireplace. They treat at the most 1½ tons an hour; but it is doubtful whether they will work well, as they have not been very successful, since a single fireplace furnace treats 24 tons in twenty-four hours, while a double fireplace furnace can only be made to treat 30 to 35 tons in twenty-four hours. Such a furnace was constructed at the Great Western, the Sulphur Bank Mine, and at the Oceanic Mine in Santa Barbara County, capable of treating 30 tons a day.

The masonry was at first made 5 ft. thick; it was afterwards increased to 7 ft., as it was found that the outside wooden braces at the Phœnix Mine were very hot, and after remaining so for a long time, the wood became partially decomposed, and took fire; 7 ft. was afterwards found to be unnecessarily thick, and the walls are now generally made 6 ft. thick. They are braced with ten large wooden beams, which are held together by iron bolts. As two furnaces are usually built together, they are strengthened by three uprights of the same size, clamped by bolts, and passing entirely around the masonry of the furnace. The cubic contents of the furnace correspond to 75 tons of ore, but as only one ton per hour is drawn from the single fireplace furnace, its capacity is one ton an hour. Every ton of ore will therefore remain three days in the furnace. When the proportion of fine ore is very large, or the ore is wet, the amount will be reduced to 12 or 16 tons a day. In the case of a large quantity of fine ore, it was proposed to remedy the difficulty by charging wood in small pieces with it, in order to keep the ore open, and allow free passages. The remedy of diminishing the width of the ore chamber answered much better.

The chamber *l* for the outlet of the vapour is underlaid with a very heavy cast iron plate to prevent any possible condensation or absorption at that point. The pipe *j* for the outlet of the gas is made of cast iron, is 18 in. in diameter and 16 ft. high where it enters the condensing chambers *b*. It projects 2 ft. from the furnace before it turns, and in the elbow there is a manhole for cleaning the pipe. Just above the outlet a cast-iron door *m* is arranged so as to observe what is passing in the interior of the furnace. On account of economy of construction two furnaces are generally built together. They are charged with ½ to ¾ medium, and ½ to ¼ of fine ore. Adobes could be used as well as any other ore, but since the introduction of the fine ore furnaces there is no necessity for making them.

The charge is introduced by means of an automatic wagon containing 17¾ cubic feet of ore, which is equivalent when moderately heaped to a ton. A cubic foot of ore at the Redington Mine weighs 115 lb. If the ore is very light the charge will weigh somewhat less. When, however, the capacity of the furnace is given in tons, that number of car-loads is intended to be expressed. The wagon is shoved by the charger to the top of the furnace where the rim of the wagon first strikes a lever attached to a wire rope which supports the cover of the furnace; this throws the furnace cover off; a projection on the bed of the wagon then throws it to one side. The workman then pulls out a pin in the wagon which discharges the ore into the furnace. As soon as the wagon is empty the weight of the furnace top and lever drives the car back, when the cover drops and closes the furnace. The weight of the charge holds the wagon in place only long enough for the car to empty itself. The workman shoves the empty car back far enough to be out of the way, covers the furnace with ashes, and prepares a new charge. One such charge is made every hour.

There are fifteen men breaking and sorting the ore. One man does the charging for two furnaces. The fuel used here and at the Manhattan Works is ordinary brushwood, which is cut by the company about a quarter of a mile from the furnace. It costs $4 a load, and is supposed to be equal to about a cord of good wood. The brushwood is used because during the summer an unlimited

supply of it can be had at a short distance from the furnace. It is much more easily cut and carted than wood, and is a cheap summer fuel. As this supply fails in the winter and teaming is more difficult, wood is then used. To use the brushwood a sheet-iron trough 3 ft. long and 2 ft. square is hung upon the fireplace door. It is exactly the size of the door and is counterbalanced, so that when not used it can be thrown out of the way. The brush is cut to the length of the trough and is packed into it with a pitchfork. The fireplace door, which is also counter-balanced, is then raised and the brush is shoved in with a fork. This brush is used directly after cutting, it flames readily and makes a very quick fire. It is charged every half-hour. When the furnace is running on fine ore it takes 1 to 1½ cords of good wood to do the work of the double furnace in twenty-four hours, or two loads of brush, which are considered as weighing a ton. When the ore is large and the circumstances favourable, the amount of wood may be reduced to one cord. The wood is a mixture of white and live oak and costs $5 a cord, delivered at the furnace. A ton of brushwood costs $5, and is sufficient to treat about 24 tons of ore.

The fireman draws the ore that has been treated and takes care of the furnace. The charge is drawn from a long narrow door *d* in the lower part of the furnace, 15 ft. below the fireplace level, but not on the same side. The iron door is raised by a tackle, and is counterpoised. The ore is drawn by a hoe and rake attached to handles of iron pipe 12 ft. to 14 ft. long. The discharge car is of iron, and is half the length of the door; it is rolled to one side of the opening, and stopped with the handle of the rake so as to be opposite the end of the door. The ore, which is black, though not cold, is drawn into it from one side of the discharge door until the wagon is half full. It is then rolled to the other side and filled from that. By the time the car is full the red-hot ore appears. The car holds one ton of fully treated ore. One load is drawn every hour from the furnace. The furnace is charged above as soon as a charge is drawn, and both are done by the same men. The average ore will weigh 115 lb. per cubic foot; after working it will not weigh more than 55 lb.

Each furnace requires six men to treat 24 tons in twenty-four

hours. A double furnace, as in Fig. 125, requires only eight men to treat 48 tons in twenty-four hours. To run four furnaces arranged as in Fig. 125 requires ten men who treat 100 tons in twenty-four hours. It requires one day's labour to treat 4, 6, or 10 tons in twenty-four hours, showing a gain of 250 per cent. on all the labour by increasing the number of furnaces to four. One engine and boiler will do all the work of four furnaces, and four to five cords of wood are sufficient to burn the ore, and keep up the steam to run the blowers.

The draught is produced by Root's blowers, shown on the plan,

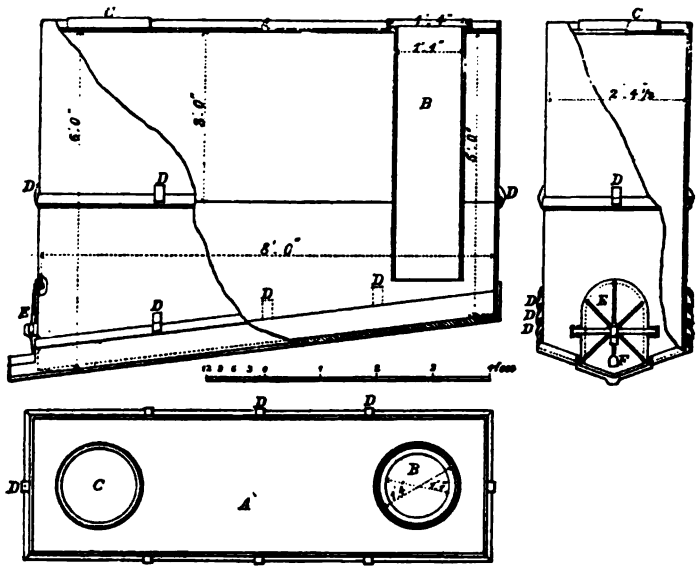


FIG. 126. THE KNOX IRON CONDENSER.

Fig. 125, at D, G, J, M, which are placed at the extremity of the condensing chambers. The bottoms of the condensers *b*, Fig. 125 and Fig. 126, are inclined at an angle of 15 deg. to 20 deg. They are joined together as is shown in the plan at the top, by means of curved iron pipes. There are sixteen of these condensers to each furnace. It is proposed, however, to have four sets of eleven, two sets of ten in one line, and six in another, making seventy-six in all, or nineteen for each furnace. They are all cast iron, 8 ft. long, 2½ ft. wide, and 5 ft. high at one end, and 6 ft. high at the other. They are set on wooden frames on a cement floor. The first condenser of the series nearest the fur-

nace is set 30 in., and the last one 37 in. above the ground Commencing with the next series, the first one furthest from the furnace is 40 in., and the last one of this series nearest to the furnace is 47 in. abcve the ground. They are made of three castings for greater ease of transportation and repairs. The top piece, which is 36 in. in depth, has sides 1½ in. deep, so as to hold the water to cool it. It is clamped to a projection on the piece below the top. When the supply of water is abundant, it is made to discharge over the top of the rim so as to run down the sides of the condensers; but when the water is scarce it is not made to overflow. At the Manhattan Works, where the first Knox furnace was built, the condensers are only half this height. They were in constant use for five years, and at the end of that time were worn out. The ore there, however, does not contain as much sulphur as at the Redington Mine.

A very small amount of quicksilver and impure sulphuric acid water commences to condense in the third condenser. It is very abundant in the eighth, and condenses through all the rest. It corrodes the ironwork more or less rapidly. It was thought at first that this action would take place chiefly on the bottom, and for this reason it was cast separate, and made several inches thick. It has, however, been found by experience, that as the bottom was entirely covered, it was in reality oxidised less rapidly than the sides, so that the bottom was afterwards only made a very little thicker than they are. It has been found that the action of the acid water is much less rapid when it is in contact with a small quantity of air. It is therefore allowed to collect for about 2 in. in the bottom of the last eight condensers, and at this height is discharged from a wooden spout into the mercury trough, and is run into the settling vats T, Fig. 125, which are 4 ft. deep and 6 ft. by 10 in. in size. In the first set of condensers, when there is a good deal of steam and but little condensed water, a wooden plug is driven into the discharge hole of the condenser, thus closing it perfectly tight. A very trifling amount of mercury is carried off by these streams when the condensed water is allowed to flow, but is collected in a tank, which shows that the condensation is very nearly perfect. The mercury commences to condense in the first to the eighth condenser, and

is discharged at once by a small opening in the bottom of the manhole door F, Fig. 126. It is collected in iron pots O, Fig. 125, which contain about an inch of water, to prevent the mercury from spattering as it falls. The iron pipe leading from the trough into the mercury pot is shaped like a syphon, so that the acid water runs off above, and the mercury collects clear. From these pots it runs into the bottling pots T, Fig. 125. The arrows outside the condensers indicate the flow of the mercury.

From the end of each set of the condensers, wooden boxes, F, H, K, N, Fig. 125, 1500 ft. long and 30 in. square, carry the smoke and fumes away from the furnaces. The boards of which they are made are clamped against a wooden frame on the inside by wooden clamps on the outside, so that it can be taken apart at any time. If nails were used they would very rapidly corrode, and the flue, if not clamped, would be likely to fall to pieces at any moment.

Three hundred feet from the furnace on the flue, there is a chimney Y, Fig. 125, which is 4 ft. square, and descends about 15 ft. It is filled with large stones over which water is made to flow from a pipe connecting with the reservoir Z. This reduces the volume of the smoke, which in this way is made quite cool at this point. One thousand feet from here the four conduits are united into one, which is continued to the top of the hill nearly a mile away. No metallic mercury is found in any part of this conduit, or in the flues leading to it, but soot deposits in small quantities. The flues are swept out once a year, and the soot so collected has yielded as high as 5½ oz. to the pound. Enough has been collected to pay for the cost of the structure, but the flues are made of this great length to carry away the smoke, which in winter collects in and around the furnace and annoys the men, and in summer to carry off the acid vapours, which would kill the vegetation. It is probable that a very large supply of water would wash down these fumes perfectly. It is quite common to see the sickly cattle in the neighbourhood collect around the mouth of this conduit, and remain there until driven away,

The engines W, Fig. 125, which do the work of the two blowers are 15 horse power; only one of the pair is used at a time; they

are also used to saw all the wood. It takes one and a half cords of wood for the engines in twenty-four hours.

All the condensers are cleaned once a week. This is done in the intervals between the charging and discharging of the furnace by taking off the man-hole door and cleaning off the sides with a hoe. It takes two or three men to take out the soot from each condenser. The soot is caught in an iron soot bucket 2 ft. by 3 ft., and 18 in. deep, which is carried on a truck. That from the first eight condensers only is rich enough to treat. That collected from the others is too poor to work, and is put directly back into the furnace with the ore. The draught into the condensers is so strong that there is no danger of mercurial vapours escaping, if not more than one is opened at a time. In order to prevent any possible accident they are only allowed to be opened one by one. After one or two have been cleaned the soot is worked up and the others are then cleaned. By doing the work in this way times can be selected when the furnace is neither charging nor discharging. It takes from Monday till Friday to clean out the condensers and work up the soot. On Saturday the conduits and spouts are cleaned, and the flasks are filled with mercury which has been collected during the week.

The quantity of soot produced is very variable, and depends upon whether the condensers have been properly cooled or not, whether the fire is quick or slow, whether the blower is run at a low or high speed, and to some extent upon the changes of the weather, so that the quantity deposited will vary at different times. The same is true of the acid water. It sometimes happens that when the blower is run at a very high speed, though everything else in the furnace is in a normal condition, but the water has given out, no acid water will be deposited in the first six or eight condensers; the heat which is usually absorbed by the evaporation of the water being so great that they remain perfectly dry.

In a charge which produces 150 flasks of mercury per week, 40 will come from the soot. The works are so arranged that two furnaces may run with one blower, or through one set of condensers, and this may become necessary when all of the blowers or one set of the condensers have to be repaired. Under such

circumstances a smaller charge has to be made in the furnace, as it is more difficult to keep up the heat. The diminution in capacity will amount to four or five car-loads of ore in twenty-four hours. Two of the furnaces ran for nearly a year without repairs, and produced from 600 to 900 flasks a month, the variation depending upon the richness of the ore.

The soot is first stirred on an inclined iron pan to separate the free mercury, and is treated with lime in retorts *e*, Fig. 125, 9 ft. long, 30 in. wide, and 18 in. high. They are so constructed that a small quantity of very hot air is introduced into them while the charge is being treated. When the charge door is opened a connection is made with the chimney, so that the draught draws the mercurial vapours away from the men. Two retorts treat 6 tons of soot per week, which contain from 40 to 50 per cent. of quicksilver. They will generally yield from 40 to 50 flasks or more. Two others would be necessary to treat all the soot, so that part of it is treated in the fine ore furnace.

A Leibig condenser *n*, Fig. 125, 10 ft. long, is added to the retort, the condensing pipe being large enough to hold at least a barrel of water at a time. Working the soot with lime in retorts is only economical when there is enough soot collected to keep the retorts steadily at work. It is very clean, and when intelligently conducted, with abundance of water for condensation, might be permitted; but it is so dangerous that it is of late years generally abandoned, and in many works the soot is collected and charged with the ore into the top of the furnaces.

At the Manhattan Works the usual method of treating the soot is in an iron kettle capable of holding about 120 gallons of water set in a frame of wood. In the bottom there is a hole into which a syphon of ½-in. gas pipe is screwed. A vertical shaft is supported over the centre of the kettle, and just below its edge, a wooden arm 4 in. by 4 in. is placed, so that in revolving it will just clear the sides of the boiler. Through this arm, iron pins 4 in. apart are run so as just to clear the bottom of the kettle. The soot, with a proper admixture of lime and ashes, is charged and the stirrer set in motion. In about twenty minutes, or a little less, all the mercury that can be mechanically separated will have gone to the bottom, and passed out through the

syphon. The material remaining behind is perfectly dry, and is at once transferred to the retort where it is treated. One man draws and treats the soot of a week's run of the furnace or the product of 150 to 175 tons of ore in one day. The experiment was made at these works of working the soot in a machine which was an iron rocker worked by steam having several motions, with an iron rake working backwards and forwards in it. The soot was mixed with lime. A very large quantity of the mercury in the soot was collected in this way by machine labour; but the difficulty of the process was that after the mercury had been collected it was again floured by the action of the rake. After a number of experiments this machine was definitely abandoned for the one described above. It is mentioned only as an illustration of how difficult it is to treat by any purely mechanical method the soot in which the greater part of the mercury is in suspension in flour.

The best men in the works at the Redington Mine are always selected for the Knox furnace. They work twelve hours, and are paid $40 and board for thirty days. If the men have families and occupy the company's cottages, they are allowed $20 as a commutation for board, but are obliged to pay $7.50 per month for rent.

The California Works.

The California Quicksilver Works also used the Knox furnace. They treated their own ores, and produced twenty-six flasks per week. The ore passes over a series of screens, and the large pieces are broken up by hand. The fine ore which is rich enough was made into adobes; one man makes nearly 900 a day. The poor fine ore was dressed by hand. Six men concentrate 10 tons in twenty-four hours, and reduce it to one-eighth of its former capacity. The dressed ore contains about 5 per cent. of cinnabar, but a considerable quantity is visible in the tailings, and it is very doubtful whether the concentrated ore pays for the labour expended upon it. The men were paid $40 a month and board.

The attempt was made here to treat the soot in the arched chamber opposite the fireplace, but the gases of the furnace contain so much free oxygen at this point that the result was the

formation of a large quantity of mercuric sulphate, and the attempt was abandoned. The works were managed with so little care that the iron condensing chambers, after having been used only eighteen months, had to be replaced. Long stalactites of acid iron sulphate hung from them in every direction, and in many cases the walls had either been eaten entirely through, or were so thin that they could be broken with the finger. It is proposed to replace the last eight condensers by brick chambers 4½ ft. high, 8 ft. wide, and 10 ft. long, with three partitions. When the iron condensers are in good repair, the soot is found from the third to the sixth; when they are out of repair it is found from the first to the third. The first six condensers are cleaned every week. Those beyond the sixth are cleaned once in three months. The amount of soot collected in the first six condensers is six barrels per week. About two cords of wood are burned in twenty-four hours. Brushwood cannot be had cheap here, and is consequently not used. Good oak wood costs $4.50 per cord. There are five men in twenty-four hours about the furnace; at night the engineer and fireman are the only workmen employed. In discharging the furnace the engineer fills the wagon, while the furnace man draws the charge down. During the day there are besides, the superintendent of the works and one man making adobes. The miners are required to deliver the ore broken and sorted at the furnace. The works in 1874 had been running for eighteen months without repairs, not having stopped more than three weeks during the whole of this time for any cause. At the end of that time they had to undergo a thorough repair.

The first Knox furnaces constructed cost between $35,000 and $40,000. They were built when wages were high, and labour was in great demand, and workmen difficult to get at any price. The furnaces built in 1873 and 1874 cost about $25,000 each, but as the prices of labour and materials diminish, they can be built for considerably less. In 1876 they cost from $18,000 dols. to $20,000, including the royalty of $5000. The first furnace built contained 170,000 bricks besides the stone. They are now built with 80,000 to 100,000. Nearly half the cost of construction was for ironwork.

Below is a statement of the cost of treating the ore in the modified Idria and Knox furnaces at the Redington Mine.

Cost per Ton of Reducing Ore in the Modified Idria Furnaces, Nos. 1 *and* 2, *at the Redington Mine, Napa Co., Cal.* 1874, *taking One Week's Run for both Furnaces.*

	$
5½ cords of wood, at $5	27.50
36 days, furnace foreman and assistant	86.00
20 ,, labourers charging furnace	38.60
12 ,, ,, discharging furnace	23.15
8 ,, ,, working soot	14.55
8 ,, fireman	14.55
9000 adobes at $5	45.00
Total cost of 200 tons of ore in two furnaces	249.35
Average cost per ton	1.25

Cost Per Ton of Reducing Ore in the Knox Furnace at the Redington Mine, Furnaces Nos. 3 *and* 4, *taking One Week's Run of both Furnaces.*

	$
42 days' labour, engineer, fireman, sootman, &c ...	81.06
17½ cords of wood at $5	87.50
1 gallon of oil for lamps...	.75
1 gallon lard oil for engine	1.20
Total cost of one week's run	170.51
Average cost per day for reducing 50 tons	24.36
,, ,, ton	.50

No account is taken of repairs during the time, as none were made. After a two years' run it was found that counting the item of repair the total cost was 90 cents a ton.

The difference between $1.25, which is the cost of treatment in the modified Idria furnace, and $0.90, the cost of treatment in the Knox furnace, is sufficiently great not only to warrant the use of much poorer ore in the latter furnace, but to settle the question of the abandonment of the old style of furnaces altogether, as they have done at the Redington Mine, as the result of this comparison.

Below is given an estimate of the principal items of expense of building one Knox quicksilver furnace in October, 1874. This cost of construction was taken from the books of the company.

Red brick	50,000 to 75,000
Firebrick	12,000

Stone for Foundation, Walls, &c.

	Weight, Pounds.
1 cast-iron lintel	3000
1 ,, draw-door frame	650
1 set draw-plates, cast-iron	3500
18 cast-iron condensers, complete, each	6500
1 firedoor and frame	650
1 flue chamber plate and door	1000
6 braces	3000
1 blower-box	2500
2 pots, &c	500
1 set grate bars	1200

Cost.

	$
1 blower	500
Blower pipes	50
1 retort and condenser	500
1000 ft. of wooden flues, cost per running foot... ...	1

1 engine with pulleys and shafting 8 in. by 12 in. cylinders.
2 boilers 12 in. by 3 in.
35 square timbers 12 in. by 12 in., 21 ft. long.
60 round iron rods 1¼ in. in diameter, 23 ft. long, 10-in. washers on both ends.
2 cars, railroad tracks, &c.

It takes six and eight masons, with their helpers, between two and three months to build one of these furnaces. The cost might be made less by making more than one-half of the condensers of wood. They might answer even nearer the furnace if there was a certainty of always having a large supply of water, so that they could be kept constantly wet.

	$
The cost of labour is about	5,000
,, building furnaces 3 and 4, Fig. 125, and condensers	60,000

Below is given Knox and Osborn's estimated cost of a quicksilver furnace, with a capacity of 75 tons, or to work 24 tons every twenty-four hours:

		$
Cast iron in furnaces, 17,000 lb. at 6 cents per pound		1020.00
,, ,, condensers, 91,600 lb. ,, ,,		5496.00
Work, planing and fitting	$90.00	
1 conical charging car	150.00	
1 discharging car	125.00	
1 Knox patent suction blower	400.00	
2 boxes borings to caulk condensers	10.00	
3 iron kettles, 120 gallons each	100.00	
Forging bolt ends, and nuts and screwing ...	70.00	
		945.00
Gross amount of foundry bill		7461.00

		$
Brought forward		7461.00
20 M. lumber at $20 per M.	400.00	
80 M. red brick at $10 per M.	800.00	
7 M. firebrick, at $60 per M.	420.00	
4000 lb. wrought iron at 5 cents per pound ...	200.00	
1 set blocks and fall (watch tackle) 6 in. ...	10.00	
2 iron sheaves 1¼ in. by 6 in.	5.00	
4 iron side pulleys	2.00	
40 ft. chain, ¼ in. wide	2.00	
10 kegs nails and spikes, at $4.50 per keg ...	45.00	
		1884.00
Labour, erecting furnace		2000.00
Boiler and engine, four horse-power		800.00
		$12,145.00

In this estimate nothing is included for grading site, drayage freight, hauling bricks, sand, clay, rock, and timber to the furnace; nor for the sheds to cover the structures. As different sized furnaces would require about the same amount, as an approximation, we may say that $30,000 would cover an extreme case, and $15,000 would suffice for a very favourable one. In a position where water for a four horse-power can be had a saving in the first cost is effected, and a greater saving in the current expenses of running.

There are two sizes of this furnace, a half-size, or 12-ton furnace, which easily treats 15 tons, and has, when crowded, worked 17 tons.

The full-size, or 24-ton furnace, crowded in the same way, will work 30 to 34 tons.

	$
The cost of half-size furnace is about	9,000
,, three-quarter size furnace is about ...	14,000

Both these sizes have been built, and gave perfect satisfaction.

The following is the estimated cost of running or operating the Knox furnace at the Manhattan Works per day of twenty-four hours when run by steam-power:

	$
For an average ore for twenty-four hours, 1¼ cords oak wood, $4.50 per cord	5.63
To men on each watch of twelve hours, or four men $50 a month and board, $2.53 a day	10.12
Oil for light and engine	.50
Cost per day of reducing 24 tons of ore	16.25
,, ton	.677

Of the 1¼ cords used, one cord is estimated for the furnace, and one-quarter for the boiler. The richer the ores are in sulphur the less the fuel required.

One of the greatest difficulties in treating the ores of mercury is the fine ore. It is too rich to be thrown away, and it must be separated from the coarse ore so as not to clog the furnace. The only way to use it until recently has been to make it into adobes, which involve a certain amount of labour. They are liable to crumble and require a considerable amount of foreign material to be mixed with them, which consequently reduces the percentage of mercury in the adobe, and necessitates a storage warehouse and a considerable supply, since they can generally only be made in the summer months. There has been a great amount of energy expended in seeking to invent a furnace that would successfully treat this kind of ore, but most of the experiments, with the exception of those made at New Almaden, have not given satisfactory results.

New Almaden.

The yard of the Hacienda, at New Almaden, Fig. 123, is cut into two parts by the Alamilos Creek, on the borders of which the spent ore is dumped. This creek is a quiet little brook in the summer, but during the season of heavy rains is powerful enough to wash out a large part of the spent ore deposited in it. One side of the yard has only furnaces Nos. 1 and 2, the other the rest of the furnaces. The two sides of the creek are connected by two bridges, one of which is on the level of the banks of the stream and the other which is 650 ft. long with an arch 100 ft. span, connects the ore bins with the furnaces Nos. 1 and 2. The plan, Fig. 123, shows the situations of all the car tracks, furnaces, ore bins, storehouses, &c. The furnaces take their numbers from the places where the old furnaces formerly stood. The Nos. 1 and 2 are the granzita furnaces. They are together on the office side of the Alamilos Creek, and have part of the old brick condensers. Nos. 3 and 8 are the tierra iron-clad furnaces with shaking grates. No. 4 was an intermittent furnace, which is now torn down. No. 5 is the first granza, no longer used. No. 6 is the old non-continuous furnace. Nos. 7 and 9 are the Exili ironclads.

The Exili Furnace.

The Exili furnace, Nos. 7 and 9, Fig. 123 and Fig. 127, is used for the coarse and rich ores called granza. It is a shaft mantle furnace, the first one of which was constructed at New Almaden in 1874 and the second in 1875. This furnace is very similar to furnaces for burning lime which have been used at Rudersdorf in Prussia since 1802. It is a continuous furnace, somewhat after the pattern of the Swedish iron ore roasting furnaces. It is heated by three

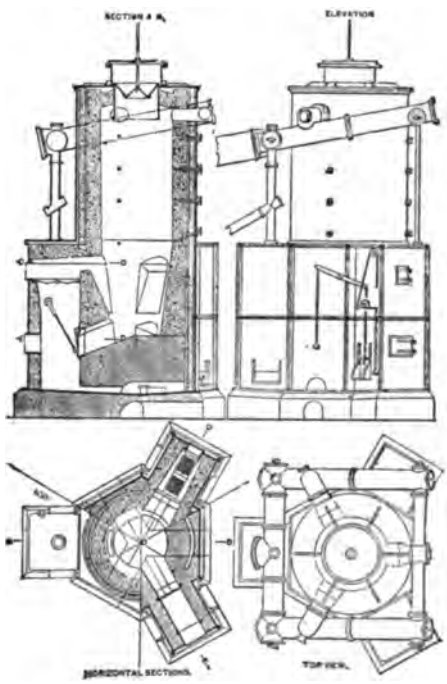


Fig. 127. The Exili iron-clad continuous shaft furnace.

fireplaces, below each of which is a cooling chamber for holding the spent ore, which is closed by a door leading from an inclined shute for withdrawing it from the furnace. The horizontal section of the furnace is hexagonal on the outside below, but round on the inside and outside above. The shaft is composed of two parts. Above the fireplace it is 11 ft. 6 in. high and 5 ft. 6 in. in diameter; below it is an inverted truncated cone,

8 ft. high and 4ft. in diameter at the bottom, making the whole interior shaft 19½ ft. high. The mantle which hermetically incloses the interior brickwork of the furnace, is made of boiler iron ⅛ in. in thickness, caulked like a boiler; against this there is a lining of red brick, a space for the expansion of the brick, and a lining of firebrick. The outside of the lower part is made of plates of cast iron well bound together and cemented with rust joints, so as to make it perfectly tight. It is lined on the fireplace side with firebrick and on the opposite side like the shaft. The lower part is made of ordinary brick set on a slightly conical plate of cast iron to collect any mercury that might collect there, and cause it to flow towards a central catch pit. The top is covered with a cast-iron plate. The ore is charged by a cup and cone inclosed in a cylindrical box with a water seal. The rod of the cone passes through a stuffing-box. The cone is counterpoised. The charge is made by lifting the cover, which is immediately closed. The ore is charged with a small amount of fuel, and is allowed to remain in the charging chamber until it has attained the temperature of the rest of the ore in the top of the furnace, before it is charged, in order to prevent any condensation taking place in the upper part of the furnace. An amount of spent ore equal to the charge is withdrawn from the lower part of the furnace, and the charge is made. When the hopper is lowered for charging the ore, it becomes filled with fumes, so that sufficient time must elapse for them to condense before charging the cup and cone again. The fumes from the furnace are removed from the chamber left above the ore at the top by means of cast-iron pipes, 12 in. in diameter; in No. 7 there are six of them, three vertical above the fireplaces and three at the top of the shaft on the fireplace side; in No. 9 there are only the three at the top of the shaft. All of these connect with the down-take, which inclines at about 10 deg. towards the condensers. No soot is likely to collect in these pipes, discs of iron attached to iron rods are so placed that they can be moved backwards and forwards from the outside in order to loosen it. The holes through which the rods pass are luted with clay when they are not in use. Around the outside of the upper part of the furnace there are twelve peep-holes at four different levels in order to observe what is going on in the furnace.

When the furnace is to be put into work it is charged with spent ore to the level of the roof of the fireplace. It is then charged with ore to which 1½ per cent. of charcoal, coal, or coke is added, up to the level of the upper peep-hole, which is 3 ft. below the level of the top of the furnace. This is done in order to have a chamber on top of the ore in which the fumes may collect. When the charging is completed the cup and cone are filled with 1600 lb. of ore, mixed with 24 lb. of coke, so as to be sure to have a proper temperature at the top of the furnace. The fireplaces are then lit. The fuel used is pine or oak; coke was formerly used as a fuel, but its use has been abandoned. The use of these fuels allows of complete control of the furnaces, but they produce more soot than the coke or charcoal. When the temperature rises to cherry red at the lower peep-hole, a car of spent ore is drawn from below, and a fresh charge added above. The charging and discharging is done every two hours, so that each furnace treats about 10 tons every twenty-four hours. Each charge remains about two days and a quarter in the furnace. The spent ore must not be immediately removed from the furnace, as it is hot and likely to give off some fumes. It must remain until it is nearly cold, or at least until there is no longer danger of fumes being given off. The volume of the spent ore differs but little from the original ore. By weight, with an ore containing 7 per cent. of mercury, the spent ore was found to be 68 per cent. The 25 per cent. being moisture, bituminous matter, and some carbonic acid.

The ore treated in these furnaces is the granza or the large and rich ore, which contains at times as much as 10 per cent., but generally varies between 5 to 8 per cent. If poorer ores than these were treated a much larger quantity than 10 tons could be put through the furnace; but with rich ores, in order to avoid loss, it is desirable to work slow. Both furnaces are worked by two men per shift, who both charge and discharge them. The expense of treatment per ton of ore is about $0.96, of which $0.40 is the cost of labour.

In 1888, No. 7 worked 345 days, treated 3185 tons of ore, produced 540,243 lb. or 7062 flasks of quicksilver from granza ores, which yielded 8.48 per cent. of mercury. No. 9 worked 157 days,

treated 1486 tons of the same sized ore, yielding 7.84 per cent., and produced 233,325 lb. or 3050 flasks of mercury.

These two furnaces stood in a line, but the disposition of the ground at the time they were built did not admit of placing their condensers regularly. As shown by the plan of the works, Fig. 123, No. 7 has two brick condensers back of it, which connect with each other to the left; connecting by two iron pipes 40 in. in diameter is another brick condenser; from this point both 7 and 9 connect with condenser tanks, Fig. 128, by three sets of iron pipes $\frac{1}{4}$ in. thick and 22 in. in diameter, and 18 ft. 10 in. long inclined at an angle of 20 deg. The condensers are iron pipes set in wooden tanks 6 ft. 6 in. wide, 10 ft. long, and 10 ft. deep. The pipes from the furnace are of sheet iron $\frac{3}{16}$ in. thick, and 22 in. in diameter. They descend into a cast-iron pipe $\frac{3}{4}$ in. thick and 22 in. in diameter, set at an angle of 20 deg., and enter a cast-iron pipe 21 in. in diameter at the bottom, inclined at an angle of 10 deg., which passes through both sides of the tank. The uprise pipe is of the same thickness and diameter as the down-take. It is 13 ft. high above the bottom one, and from its top the exit pipe of sheet iron, $\frac{3}{16}$ in. thick and 22 in. in diameter, passes out at the same angle as the entering one. Both the entrance and the exit pipes have hand-holes for the convenience of cleaning them. Both the upright and the bottom pipes have heads which are luted and bolted on, which can be removed when desirable. The tank is kept full to within 6 in. of the top with water, which is kept by a supply pipe at a constant temperature. The lower end of the bottom pipe is pierced by a small tube to let out the mercury and the condensed acid. This flows over a wooden shelf which, up to the present time, has always been covered with asphalte, but will in the future be covered with rubber. This shelf is used for collecting the soot. Each tank contains three of these inverted syphons. This system answers so well that all the wood and glass condensers which formerly occupied the larger part of the space back of these furnaces have now been abandoned. Each pipe enters the tank, and continues its course after having passed through it to a brick condenser. These brick condensers are coated on the inside with asphalte. Instead of having brick partition walls, they are

provided with three "revolving screens," placed one above the other, but so that their position can be changed like stove-pipe dampers by moving the axes which project through the side

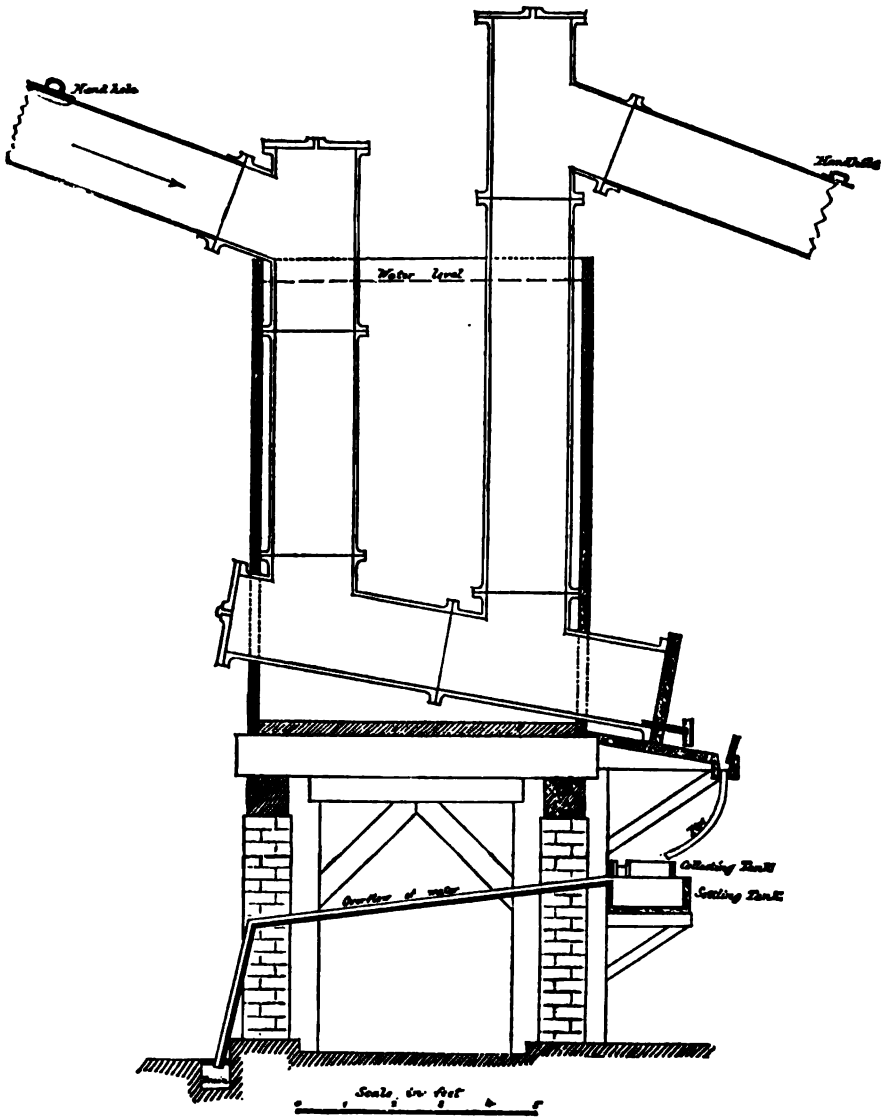


FIG. 128. IRON TANK CONDENSER FOR THE EXILI FURNACES.

walls. To these axes damper blades and wooden projections are placed, which are painted with coal-tar. When the damper is placed horizontal, the passage across the condenser is closed.

When they are all vertical the chamber is divided into compartments. By setting them at irregular angles the dampers make the current of gas take a variety of motions which are all favourable to the condensation of the mercury. The conduits then unite, and after passing two more brick condensers, common to each, the gases pass by a wooden flue, 21 ft. long, into a tower in which there was formerly a fireplace for use in hot weather, now abandoned, to a cylindrical brick flue 4 ft. in diameter and 400 ft. long, to another chimney 60 ft. high, 5 ft. at the base, and 4 ft. at the top. This flue enters the chimney 200 ft. above the charging floor of the furnace, and 235 ft. above the floor of the works. The first tower has recently been connected by a wooden flue 210 ft. long with the flues for 3, 6, and 8, which connect with the Guibal

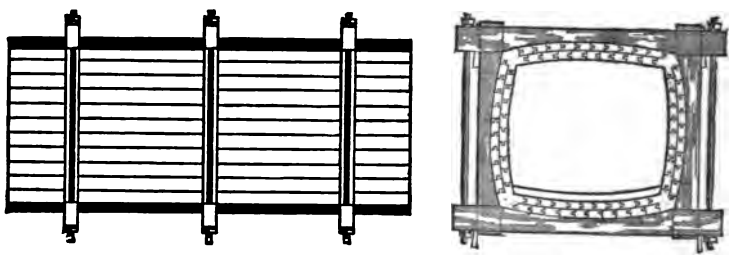


FIG. 129. WOODEN FLUE AT NEW ALMADEN.

fan on the slope of the hill. In this way these fumes pass through nearly 900 ft. of flue before reaching the chimney.

The wooden flues which are used for all the furnaces on the office side of Alamilos Creek are constructed as shown in Fig. 129. The flue is built of two thicknesses of redwood boards, 1 in. wide, tongued and grooved, and arranged to make joints. To insure of the flue being gas-tight asphaltum paper is placed between the upper and lower boards. As originally constructed, corner pieces, as shown in Fig. 129, were used. These have since been abandoned. The corners are now made of the boards themselves. To insure the greatest resistance they are made slightly curved, having a rise of 2 in. in 3 ft. To hold them together they are supported every 6 ft. by wooden frames of 6 in. by 3 in., the upper and lower ends of which are cut out on the inside to the shape of the flue, and tied together by iron rods. The sides are mortised into the top and bottom. They are cut out both on the top and

bottom and they are wedged up against the flue with wooden wedges. When finished they are coated on the inside with asphalte, and are painted on the outside. With this construction these flues are perfectly tight, and generally remain so; but in case they should contract, the nuts may be screwed up and the wedges driven, so as to insure of their being made tight again. As, however, they are made when the timber is quite dry, and the gases which pass through them are quite moist, they usually swell tight and remain so, and require but little looking after after a period of disuse.

There was formerly connected with these furnaces an iron condenser invented in 1873 by Mr. Fredler. This was a cast-iron box 10 ft. 6 in. long and 5 ft. 6 in. wide and high, with a sloping roof. In this box, in the direction of its length, were three hollow partitions for the use of water which divided the condenser into four compartments, each one of which had a sloping bottom and goose necks. The two end partitions were open at the top. The middle one was interior. The three were connected by pipes, so that the water went from one to the other. The water pipe was on top, and not only supplied the hollow partitions, but made a spray of water over the roof. It was very effective, but was abandoned on account of the obstruction of the gas, and because it was badly attacked by the cooled acid fumes.

The Guibal fan, which furnishes the draught for furnaces Nos. 3, 6, 7, 8, and 9, is 8 ft. in diameter and 28 in. wide, with wooden blades. The frame and box of the fan, except the central part, which surrounds the shaft, which is made of cast iron, is made of wrought iron, which has to be replaced every two years. The floats are made of boards which were formerly soaked in paraffine, but it has been found by experience that they answer equally well when they are only painted with paraffine paint. The use of the fan has saved so much in fuel, that it has been adopted on the flues of furnaces Nos. 1 and 2, so that the auxiliary fires are no longer necessary. It was formerly run by a small Knight's wheel with water from the mine, which had to be pumped over the hill. It is now run by a 6 horse-power engine, which also pumps the water used for the elevators and for condensation into a tank 80 ft. above the furnaces.

The Hüttner-Scott Tierras Furnaces.

Notwithstanding the success of the Exili furnaces there still remained the very large quantity of tierras ores, which amounted to two-thirds of all the ores produced, which had either to be made into adobes or left untouched, which could not be thought of, as they comprised such a considerable part of the production of the mine. Taking the idea of the Hassenclever furnace, the first tierras furnace was invented in 1875 and patented in 1876. It was called No. 5. It still retains its place on the map, but is no longer used, as the larger furnaces are found to work more economically. It consisted of two separate shafts, back to back in the direction of their length. Each of these chambers was filled with tiles inclined at an angle of 40 deg., the width of the shaft, and overlapping each other. Under each brick is an opening into a chimney built into the side walls, and common to the two furnaces. These are divided one at one-third the height of the furnace and the other at two-thirds. The fireplace is placed in the lower one, so that the flames from it are obliged to go but once over the ore and through the furnace before finding their exit. Both sides are charged at the top from the same hopper and discharged from below from the same outlet. The heat when the furnace is filled with ore traverses the openings underneath the tiles, and when it is extracted the ore is discharged from the lower part into an iron wagon. This causes a movement in all the ore of the furnace, which descends in the shaft, turning over as it falls, and exposing fresh surfaces, and comes to rest again when the discharge opening is closed. This furnace roasted the ore quite well, but the vapours which escaped were quite hot, and the treatment required a large amount of fuel. The capacity of the first furnace was only 6 tons in twenty-four hours; it was afterwards doubled when it became apparent that much greater economy would result from the construction of a larger furnace. The cost of treatment in the 6-ton furnace was found to be $8.23 per flask and $3.13 per ton of ore treated, which was reduced nearly one-half when the ore was washed on a large scale.

The success obtained with No. 5 led to the construction in 1876 of a double furnace, each side of which had a capacity of 12 tons or for the furnace of 24 tons in twenty-four hours. This double construction was adopted because, as it had been shown that larger furnaces were more advantageous, it was hoped that repairs to one could be made while the other was running, but experience has shown that the heat is too great to admit of it. The double furnace, Fig. 130, was called No. 8. It is shown on the plan of the works, Fig. 123. It was built next to No. 6, and has done good work ever since. It really is two furnaces, like No. 5, placed side by side, instead of back to back as before. The ground plan of the two is 37 ft. long and 9 ft. 6 in. wide.

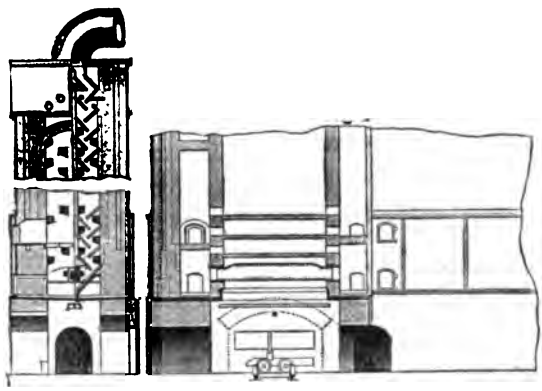


FIG. 130. THE TIERRAS FURNACE NO. 8.

To reduce the amount of fuel consumed arches are placed in the hollow walls, so as to force the vapours to encircle the furnace three times, or, in other words, to pass four times over the ore before they escape from the furnace. All the air which enters the fireplace is heated by passing it through the spent ore, which heats the air and reduces the temperature of the ore, thus saving fuel. The products of combustion at their highest temperature pass through the nearly spent ore, and then gradually grow cooler by coming in contact with the cold ore, and so leave the furnace at a comparatively low temperature. These changes produced a very notable reduction in the cost of the treatment. The ore chambers of this furnace are much larger than No. 5. They are 18 in. wide, 9 ft. long, and 31 ft. high. The tiles are set at an angle of 45 deg., so that the upper one is at

right angles to the one below it. The last of these, as it comes directly in contact with the flames of the fireplace, is made of cast iron. The fireplace is in front, and is 3 ft. above the bottom of the ore chamber. This space is utilised to cool the spent ore. The bottom of the chamber is 6 ft. above the furnace level, making the total height of the furnace 37 ft. The wagons which carry the ore to all the tierras furnaces are covered with a screen made of 1-in. iron $\frac{3}{16}$ in. thick, set on edge, and bars with $\frac{1}{4}$-in. round iron $1\frac{1}{2}$ in. apart, and kept in place by pieces of pipe between them. This is fastened permanently to the top, so that it cannot be removed except by unscrewing it, so that no ore except of this size can enter the wagon. The ore is charged into the furnace by V shaped compartments which are opened and closed by a lever. The slides at their bottom are usually single. In this furnace there are three of them. It is discharged by a shaking table, which is moved by bars outside the furnace, and which close both sides of the discharge when at rest. As soon as the spent ore or "slag car" is run into the spent ore chamber below, the hanging doors in front of them are closed to avoid the dust. The width of No. 8 is so great that two slag cars are run into the spent ore chamber at a time. The motion given to the lever discharges the ore from one side and the other into the car. The men know by long habit the number of oscillations to give to the lever in order to fill the car with the spent ore. The furnace has a capacity of 1 ton per hour. As when full it holds 32 tons, each charge remains thirty-two hours in the furnace. The charge in each hopper is 1000 lb. per hour. They are charged alternately every half-hour, so that the total charge is 1 ton per hour. The discharge is made every quarter of an hour. Two men do the whole work of the furnace. When first constructed the cost of treatment per flask was $4.69, and the cost per ton of ore $1.73. The present cost per ton of ore is about half. In the year 1888 this furnace worked 200 days on tierras ores, yielding 1.36 per cent. of mercury, in which time 4737 tons of ore were treated, producing 128,979 lb., or 1686 flasks of quicksilver.

The condensing system of this furnace consists of brick chambers connected with the furnace by two diverging pipes.

The first one has three chambers, and is water backed. This is connected by six ironstone pipes 22 in. in diameter with two wooden water tank condensers placed side by side, but 18 in. apart, like Fig. 128, but much longer; three of the pipes enter each tank at an angle of 10 deg. They enter a wooden box 2 ft. 4 in. square on the inside, constructed like Fig. 129, but divided in the middle, so as to make two tight compartments; on the top of each of these compartments is a hand-hole for cleaning it. From the bottom of the first one an ironstone pipe, 22 in. in diameter, descends to a wooden flue, like Fig. 129, 24 in. square on the inside, inclined at an angle of 10 deg., passing through to the outside, like Fig. 128. The uprise pipe is of ironstone also. It enters its wooden flue above, from which an ironstone pipe inclined up at an angle of 10 deg., passes out. Each tank is built of 3-in. planks 10 ft. long, 13 ft. 6 in. and 14 ft. high, set 3 ft. 6 in. above the ground, with the water line 4 ft. 6 in. from the top. These are connected by six earthen pipes, with a brick condenser of two chambers, and these by pipes 25 ft. long with a chamber of two condensers. The last condenser connects with the flue system by two divergent pipes, 27 ft. long, which run together and connect with a wooden flue, 75 ft. long, which discharges into a brick tower, and enters a brick flue, which itself discharges to the flue in which the Guibal fan is placed, into which the flues of Nos. 6, 3, 7, and 9 also discharge.

The success obtained with No. 8 led to the erection of No. 3, which was started in 1878. It is of the same general type as Nos. 5 and 8, but much larger. It is an iron-clad furnace, being protected on the outside with heavy iron plates bolted and cemented together with cast joints, and supported by bracing as shown in Fig. 131. Peep-holes are arranged in the mantel in order to see the condition of the ore at different levels in the furnace. It was built with the expectation that, with the slits in the middle compartment, which are only 3 in. in width, it might be used for tierras, and as the others were made 5 in. they might be used for granzita ores, but it was found on trying it that the heat, which was quite sufficient for tierras ores, was not high enough for the granzita, so that the furnace is now only used for tierras ores. It differs somewhat in construction from No. 8

from the fact that there are three sets of ore chambers in each of the double furnaces. The tiles used for the shelving are made in New York and New Jersey. Those for the outside chambers, which have a 5-in. slit, and were intended for granzita ores, are 26 in. by 31 in., and 2½ in. thick. They wear but little by abrasion, and do not break when they are of good quality. In the granzita furnaces at the point where they lap about 3 in., they wear down to a knife edge in three to four years, and have to be replaced. In the other furnaces there is little or no wear. In the upper part of the furnace, when they are cool, notwithstanding the constant abrasion from the sliding of the ore over

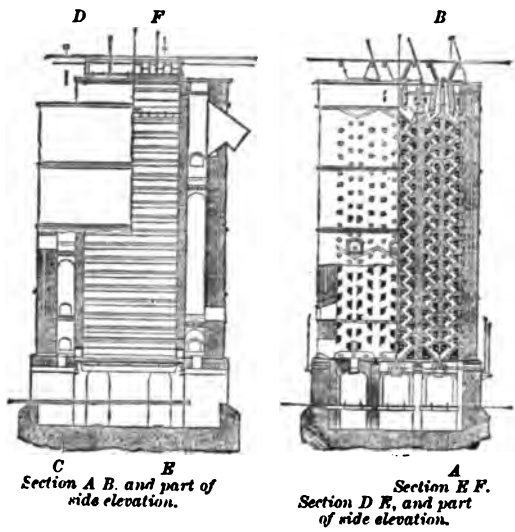


FIG. 131. THE TIERRAS FURNACE. No. 3.

them, they last several years; but in the lower part, where they are exposed to the direct heat, they first become brittle and crumble, and have to be replaced. They wear out in the granzita much quicker than in the tierras furnaces. On the back edge of each tile there formerly was a shoulder set at 45 deg., by which it was fastened into the wall of the furnace. The tiles of opposite chambers are set against each other, shaped bricks being fitted under and over them. There are four rows of firebrick, and then another set of shelves, and so on. The tiles are now made as perfectly plain slabs, as it was found that the shoulder made them more expensive, and did not answer any

better than the plain ones, and that on account of their variable thickness they crack at the back, so that the use of shaped bricks has been entirely abandoned except for No. 3, for which there is a considerable supply still on hand. The plain bricks are set in slots left in the masonry for the purpose. The underside is supported by a brick projecting from the back of the furnace. The wall which separates the two chambers of the same compartment is 9¼ in. thick, that which separates two compartments is 13¾ in. thick. Each chamber is 23 in. wide, and the tiles are 21 in. apart vertically. The hollow walls used in constructing this furnace were found to be unsuited for mercury furnaces, and have been abandoned in the subsequent constructions. The fireplace is placed 5 ft. above the discharge opening. Air is drawn in through the spent ore to cool it down so that it is discharged nearly cool into the wagons in the spent ore discharge chamber underneath. The capacity of the furnace is 36 tons in twenty-four hours. It is charged at the top by slides moved by levers in such a way that no fumes can escape. The furnace is discharged through openings made in the cast-iron plates which cover the pillars of the spent ore chambers. This opening for the outside chamber is 5 in., and for the others 3 in. wide, and the whole length of the furnace. Below these openings is the discharge valve which closes it. It is made of cast iron, and is flat on the top, but is reinforced on the bottom for greater strength. For the outside compartment it is 15 in. wide, and for the other 9 in. It is supported on wheels running on iron rails, and is moved by levers placed on the outside of the furnace, as shown in Fig. 131. In this furnace there are three of them, two with two slots and one with one. When the valve is in the centre under the opening the ore comes down on it on both sides and rests there, but when it is moved backwards and forwards by the lever the ore falls from one side and the other in a stream into the car below. When it is replaced in the central position the ore remains at the angle of its friction at its edge. The discharge of ore is made every ten to fifteen minutes, so that about one ton of ore is discharged every two hours. A ton of fresh ore is then added in the ore hoppers above. As the furnace holds 51 tons of ore, each charge remains thirty-four hours in the furnace. Wood is used as fuel, and 2½ cords of

it are consumed in twenty-four hours. The work of the furnace requires three men on each shift. Two of these men are paid $2.50, and one $1.25. The cost of extraction per ton of ore is $0.72. In 1888, No. 3 worked 255 days, treated 9111 tons of tierras ores, yielding 1.29 per cent., and produced 235,467 lb., or 3078 flasks, of quicksilver.

The furnace connects by three iron pipes with the condensing system. The first condenser also connects with three low brick condensers, also connected with a shelf drying kiln, Fig. 132, by two iron pipes 30 in. in diameter and 70 ft. long, made of $\frac{3}{16}$-iron, which returns its fumes into the end of the first brick condenser. The main condenser is of brick, and was originally composed of fifteen compartments, Nos. 5, 8, and 11 of which have been cut out, which makes four separate condensers in a straight line directly behind the furnace. The connecting pipes are all cast iron, and are 16 in. in diameter; 4 connects with 6 by 8 pipes, 7 with 9 by 8 pipes, 10 with 12 by 9 pipes, 15 connects with a brick tower, after which it joins the wooden flue coming from the furnaces 7 and 9.

As these ores contain considerable moisture which it is desirable to dry out, the kiln, Fig. 132, on the same general principle as the furnaces, was introduced. It answers its purpose much better than that of furnace No. 6, and serves at the same time as a condenser. The fumes come from the first brick condensers by two inclined pipes, 60 ft. long, as shown in the plan, Fig. 123. The kiln, as shown in the sections AA, BB, and CC, is composed of two vertical compartments, in each of which there are twelve shelves resting on iron supports, placed at an angle of 45 deg. to each other. These chambers are surrounded on the outside by compartments the whole length of the kiln, around which the hot gases circulate freely, so that a considerable part of their heat, as the walls are very thin, is taken up by the ore The moisture escapes from openings made in the hollow walls of the kilns. The ore is drawn from the ports below, and descends just as it does in the furnaces. In order to provide for any possible elevation of the heat, and consequent loss of mercury, inclined iron plates are put into the walls and drain chambers on the foundation. The use of this kiln has considerably reduced the cost of working these ores.

From the dry kiln the fumes pass back again through three low brick condensers, then through chambers arranged two and four together, then into a wooden flue, 81 ft. long, to a brick tower, then through 51 ft. of flue, and joins the common flue leading to the Guibal fan on the side of the hill common to all the furnaces on this side, then by a brick flue 600 ft. long to a chimney 5 ft. square at the base and 4 ft. at the top, which is 385 ft. above the

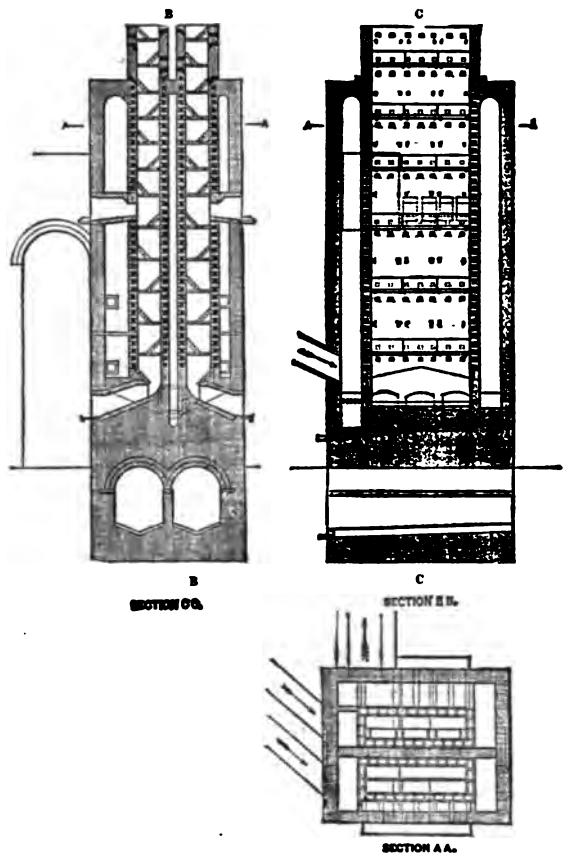


FIG. 132. THE SHELF DRYING CHAMBER.

floor of the works. The length of the flues of Nos. 3, 6, and 8, is nearly 900 ft.

A sufficient number of furnaces having been supplied for the fine ores attention was turned to the granzita, or medium-sized poor ores, and in 1879 furnace No. 2, Fig. 133, with two ore chambers, and in 1880 furnace No. 1, Fig. 136, with two pairs of

them, were built on the opposite side of Alamilos Creek, and was connected by a wooden bridge having an arch of 100 ft. span with the ore bins on the opposite side.

Furnace No. 2, Fig. 133, is very similar in construction to furnace No. 8, Fig. 130, but differs somewhat in the details of its construction. The outside masonry is 36 ft. high and 12 ft. 6 in. at the base. The interior is composed of two ore compartments, each one of which is 27 ft. 6 in. high, 2 ft. 6 in. wide, and 11 ft.

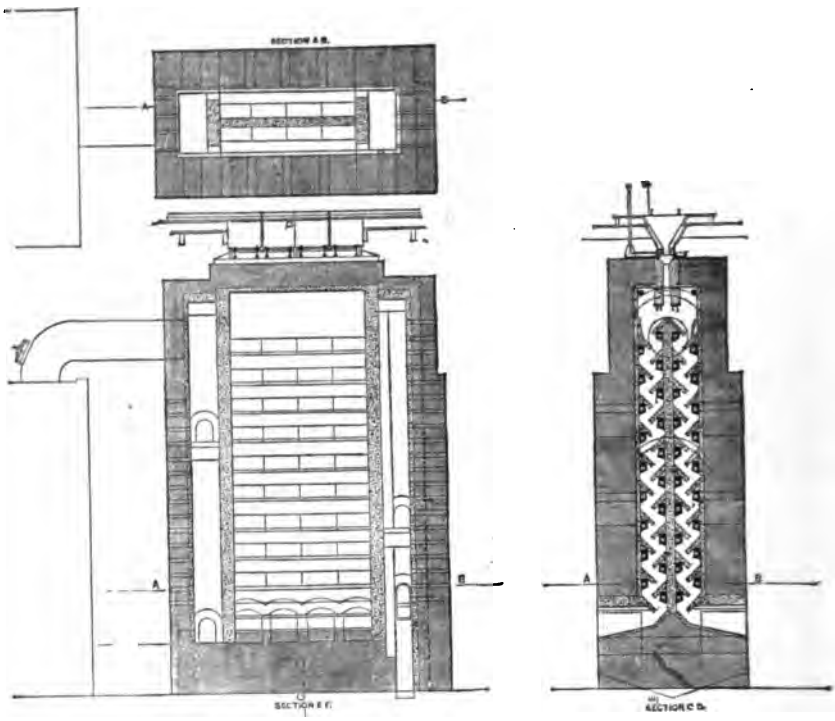


FIG. 133. THE GRANZITA FURNACE NO. 2.

6 in. long. Each side of each compartment has ten tiles set at an angle of 45 deg. to each other, with an opening of 7 in. between them. The wear of these tiles is on the upper surface; they are destroyed more by cracking than by abrasion. The chambers for the escape of the gases are 2 ft. 6 in. wide, and so arched that the heat makes three turns round the furnace before it escapes into the first brick condenser. There are two fireplaces in which wood is used. The furnace is charged by a single hopper, which

delivers the ore to both sides, but in order to secure perfect regularity in the charging, they are divided in their lengths into several compartments. The ore is charged in both Nos. 1 and 2 from wagons covered with a grating $3\frac{3}{4}$ in. by $2\frac{1}{2}$ in., made of $\frac{1}{2}$-in. iron $\frac{3}{8}$ in. thick, with half-round iron, screwed in place like the other cars. These wagons are dumped on to the V charging holes in the top of the furnace. These have sliding valves moved by levers which discharge the ore into the furnace, and the valves are closed. To prevent any escape of fumes, the valves are at once covered with fine ore. If any soot is used in the charge, it is put on separately and immediately covered with ore so as to prevent dust. The ore is raised from the ground level on a water elevator, the collecting water tank of which serves as a tank for washing the soot. There is a piston elevator here, but it is not used. The spent ore is removed from the two draw pits at each end of the furnace, about 150 lb. being drawn at a time, removing 1000 lb. of roasted ore. When this has been done 1000 lb. of fresh ore is charged, and the hoppers again filled. This is repeated at intervals of forty minutes, so the furnace treats 18 tons in twenty-four hours, and consumes about three-quarters of a cord of wood in the same time. The furnace requires three men per shift, or six men in twenty-four hours. There is one man above to charge the ore, and two men below to draw it and keep up the fire. When No. 1 is running, as the two furnaces are close together, the same men attend to both furnaces. Four men besides are required to bring the ore to the furnaces from the other side of the creek. The furnace is of limited capacity, and nothing has been more clearly proven at New Almaden than that the economy increases directly with the capacity of the furnace, so that small furnaces will no longer be built. The cost of treating a ton of ore is \$1, \$0.46 of which is for labour and the rest for fuel. In 1888 this furnace worked 128 days on granza and granzita ores yielding 1.995 per cent., treated 2375 tons of ore, and produced 94,783.5 lb., or 1239 flasks, of quicksilver.

The condensing system of these furnaces is made up of the old brick condensers and glass houses. The brick condensers are usually 27 ft. high and 6 ft. on the outside. This system,

which forms part of the condensation apparatus, is most extensively carried out in connection with this furnace. It formerly had twenty-two chambers, but Nos. 3, 5, 8, 10, 12, 14, 16, and 19 have been cut out, and No. 20 has been cut in two in the middle, making two; and the ten that remain are single or double as it was found best to make them. The top of the intervening spaces, which are 4 ft. wide, are connected by brick arches. The connection between two chambers is made by pipes of iron near the ore furnace, and afterwards by earthen pipes. The Table below shows their arrangement:

With the furnace	1 iron pipe 42 in.
,, Nos. 2 and 4	3 ,, pipes 24 in.
,, ,, 4 ,, 6	4 ,, ,, 16 ,,
,, ,, 7 ,, 9	4 earthen pipes 18 in.
,, ,, 9 ,, 11	3 ,, ,, 18 ,,
,, ,, 11 ,, 3	6 ,, ,, 18 ,,
,, ,, 13 ,, 15	6 ,, ,, 18 ,,
,, ,, 15 ,, 17	6 ,, ,, 18 ,,
,, ,, 18 ,, 20	2 wooden ,, 24 sq. in.
,, ,, 20 ,, 22	2 ,, ,, 24 ,,

Most of these pipes have hand-holes in them to remove the soot without disconnecting them from the chambers.

A plan elevation and sections of the chambers are shown in Fig. 134. As shown in the section E, F, G, H, each chamber is a tall narrow structure divided into two compartments, and having openings 2 ft. square at the top and bottom to facilitate cleaning them. In those where the heat is the greatest, these openings are closed with iron plates 28 in. by 30 in., when the furnace is running. They are held in position by wooden braces across the front, and luted with a mixture of clay and ashes. In the cool ones they are closed with a window sash, as the acid waters would attack the iron. The floor of each condenser slopes two ways, towards the centre and towards the sides, so as to deliver at once any of the condensed mercury as well as the acid waters to the sides. The floors are constructed with the greatest care, and are then coated with cement, over which in all the chambers which are far enough away from the furnace to admit of it, a thick coating of asphalte is placed and carefully smoothed down.

As the walls are constantly exposed to hot acid fumes, they finally disintegrated and caused much trouble from leakage. In order to provide against this and make them light the walls of those condensers which are cool enough are now covered with a mixture of asphalte and coal tar put on hot. This answers the purpose perfectly on new walls, but on old walls, already partially disintegrated, it is difficult to make it adhere. In front of each system, and on both sides of it, gutters 18 in. deep and 9 in. wide, are placed sufficiently inclined to carry all the

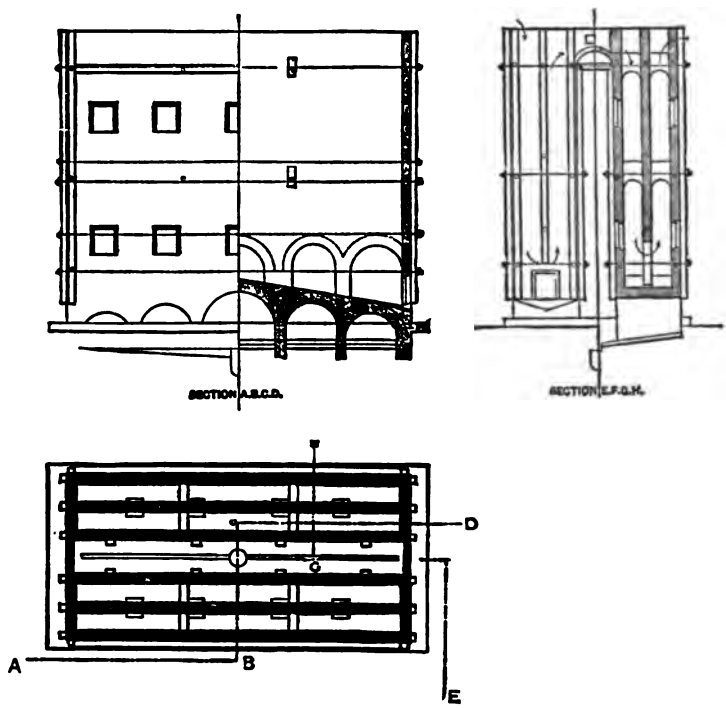


FIG. 134. BRICK CONDENSERS.

mercury and condensed acid to the receiving vats. This gutter is lined with asphalte to keep it tight and to prevent the action of the acid waters.

The earlier forms of these condensers were built without the intervening air spaces, and answered perfectly so long as the furnaces were non-continuous, as each chamber had time to cool off; but when they were made continuous the accumulated heat prevented the condensation in the first condensers, and they were

then altered as they now are, but even then it was found that there was considerable loss, and glass houses were built to supplement the work. These were adapted at first to all the furnaces, but have been entirely abandoned except in connection with furnaces Nos. 1 and 2. They answered extremely well until the idea of the water back, now used in connection with the first chambers of all the condensers, was introduced. This is used in chambers 1 and 2 of this furnace, and will be described under furnace No. 1. With the most recent construction the brick condensers seem to answer better than any others. When the furnaces are hot they do not suffer from corrosion, and when they are cool they are protected by the asphalte coating. At one time they seemed likely to be altogether abandoned, but

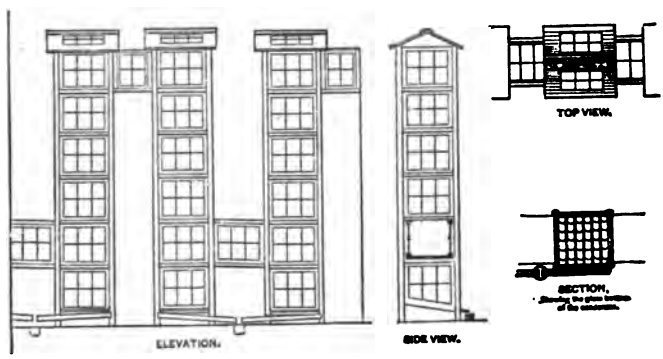


FIG. 135. GLASS CONDENSERS.

they are still in use, and are likely to be. Most of the mercury, however, is collected in the water back chambers.

The glass condensers are shown in Fig. 135. These are separate chambers constructed entirely of wood and glass, without iron nails of any kind. They are 25 ft. high and 4 ft. square, and are covered with a glass roof with a pitch of 25 deg. The bottom is made of glass, cut V shaped and lapping. This is inclined towards the outside, and connects with the drain trough. These chambers connect at the top and bottom alternately, but the bottom connection is 4 ft. above the bottom of the condenser. But little mercury is caught in them, and this is collected in an earthen pot, which is placed directly in front of each chamber at the end of the runners. What is collected is mostly acid water.

To f

FRONT OF FURNACE

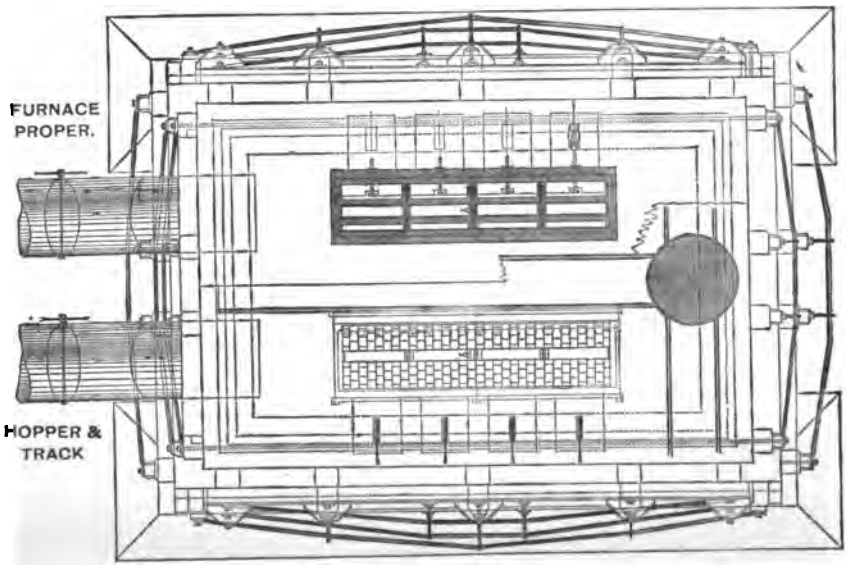
RIGHT VIEW

SECTION THROUGH SECOND ORECHAMBER

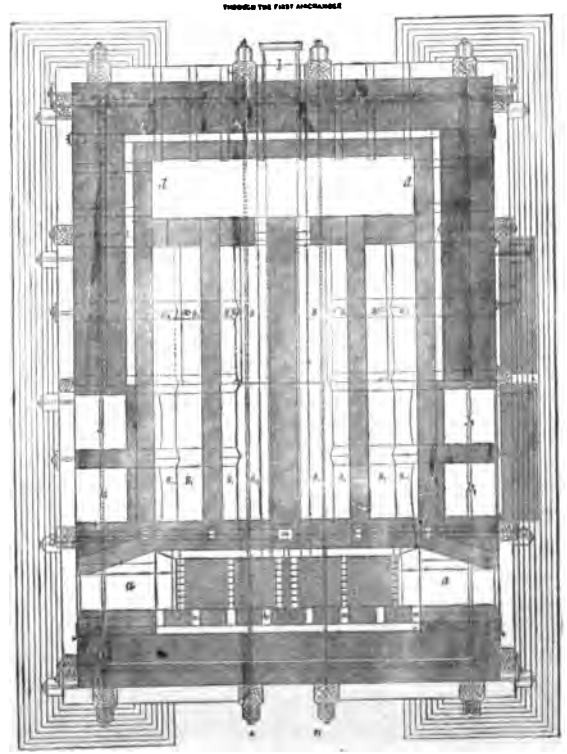
VERTICAL SECTIONS

THROUGH THE FIREPLACE

THROUGH THE ORECHAMBER



HORIZONTAL SECTIONS
THROUGH THE FIRST ARCHAMBER



These condensers answer perfectly, so long as they are in constant use, for the moisture in the fumes keeps them tight. The temperature in them varies from 55 to 80 deg. F. When the furnaces are not kept running they contract, break, and interfere with the draught. If they were to be extensively used the wood might be boiled in paraffine, as is done in some of the chlorinating vats, but this would add to the expense. After passing twelve of these condensers the fumes go by a wooden flue to a brick tower and from thence to a chimney on the top of the hill, at the foot of which is a Guibal fan.

The last furnace constructed at New Almaden is the granzita furnace No. 1, Fig. 136, built in 1880, which is on the whole the most interesting furnace at New Almaden. It consists of a double furnace 36 ft. high, 25 ft. 6 in. long, and 17 ft. 6 in. wide. The foundations are 7 ft. deep, and contain an arch, the bottom of which is covered with iron plates *o*, to intercept the flow of any escaping mercury. Each ore chamber is 27 ft. high, 11 ft. 6 in. long, and 26 in. wide. It has nine series of tiles placed as in the others, at an angle of 45 deg. These are placed 30 in. apart vertically on each wall, 15 in. above those on the opposite wall, and at right angles to them. The tiles are 36 in. by 16 in. by 3 in. They are supported at the top by projections from the side walls, and at the bottom on firebricks, resting in depressions 1 in. deep in the tile below, so placed as to present an edge to the descending column of ore. As the bricks are 9 in. long an opening 8 in. wide is thus left between the tiles. The outside chambers are so constructed as to make the heat pass several times round the furnaces. At the bottom of both the upper and lower chambers *e e*, is a door *g*, bricked up while the furnace is working, but opened at the end of the run, to clean out the dust which has accumulated there. The fireplace *a a* is made for burning wood, and is charged from both ends. It is fed with warm air by a pipe *l*, which is 16 in. in diameter, which passes through the condensers, and also receives a supply of warm air through the openings *m*, built in the walls of the fireplace and furnace. The openings *c* allow the vapours to enter the chambers *e*. In the combustion chamber wall these openings are horizontal, but in the vapour chamber they are inclined with the tiles; but as dust

is always carried with the gases they are slanted into the vapour chamber as well, so that any accumulation of it will be collected there, from which, at the end of the campaign, it may be removed by the doors *g*. In case there should be any obstruction either on the tiles or these openings, a series of circular peep-holes *x*, and made of 3-in. gas-pipe, are arranged in the end walls, through which an iron bar can be introduced to remove it. These holes are ordinarily closed with clay. The furnace is charged by the hoppers *h*, of which there are four on each side worked by levers; the turntable *t* allowing the ore car to be brought close to them. The discharge for the spent ore on the front side *j* is made on to an iron apron; from the back side it discharges on a lower level *k*. Both these compartments are closed with counterbalanced iron doors. The discharge from *k* is made to fall into an iron car. There are four of these double doors on each side of the furnace.

The heat from the fireplace *a* passes through the lower third of the ore chamber into the chambers *d d* shown in the front view, then through the middle of the ore chamber into the compartment *e e* shown in the section to the right, then through the upper part of the ore chamber into *f f*, and then through the iron pipes *i* to the condensers. These pipes are provided with valves to regulate the draught. In order to prevent any obstruction of the ore column from damp ore baking on the upper tiles, ore rakes *r* are introduced, which are controlled by a handle on the top of the furnace. These rakes reach the upper tile, and a few motions of the handle loosens any ore that may have become attached there.

The ore is drawn from either side of the furnace alternately every forty minutes, two tons by volume being taken every eighty minutes. To effect this 250 lb. are taken from each of the eight draw-pits on each side of the furnace. As soon as this is done two tons of fresh ore are charged from the hopper, and the hopper filled at once. Thus the same amount of ore is charged as is withdrawn. The firing is done every hour; the fuel is oak. The fireplace must be kept at cherry red, which takes from $1\frac{1}{2}$ to $1\frac{3}{4}$ cords of wood in twenty-four hours. The furnace when full holds 45 tons. It treats 36 tons in twenty-four hours, each

charge remaining thirty hours in the furnace. The ore treated in No. 1 and 2 is frequently not over 1 per cent. and sometimes falls as low as ¾ per cent. The cost of the operation is $0.64 per ton of ore, of which $0.28 is for labour. This is an excellent illustration of the economy of working with large furnaces.

When the furnace is worked alone, one charger and two slag and firemen are required. The fire and slagmen pile the fuel, send up the ore by the elevator, and help charging the bottles with quicksilver, and screwing the corks into them. When No. 2 is working three men do the work of that furnace as well. In addition to this labour, two men are required part of the time to bring the ore from the ore bins.

In 1888 this furnace worked 236 days on granza and granzita ores, yielding 1.395 per cent, treated 8420 tons of ore, and produced 235,084.5 lb., or 3073 flasks of quicksilver.

The system of condensation employed for this furnace is brick condensers and glass houses. There were formerly fifteen brick condensers in a single structure, but Nos. 3, 5, 8, 11, and 14 have been cut out and the connections made by pipes commencing at the furnaces. The first pipes connecting with the condensers are two of iron 4½ in. in diameter. In the condensers, between 2 and 4, four iron, 28 in., and four earthen, 20 in.; between 4 and 6, four earthen, 18 in., one iron, 20 in.; between 7 and 9, seven earthen, 18 in.; between 10 and 12, eight earthen, 18 in.; and between 13 and 15, eight earthen of 18 in. In this furnace there are three chambers, 1, 2, and 4, which have water backs. In furnace No. 2, Nos. 1 and 2 only are water backed. These are cast-iron boxes with outside flanges, set into the masonry, with a cover which is lined with wood bolted on to it, which is easily removed to examine the interior. At one side pipes running the whole width of the furnace open into the chamber, and on the other there is a partition to force the circulation of the water. This chamber is 43 in. long, 16 in. high, and 14 in. deep on the inside, and has a flange of 8 in. for the bolts. Into the bottom of this 4-in. wrought-iron pipes, ¼ in. thick, are screwed in two rows, six above and four below. The ends of the pipe project 5 in. These pipes run across the whole condenser. The pipes are coated with tar. This is made of a mixture of

asphalte and coal tar boiled together until a piece of paper dipped into it and plunged into water shows the tar still soft. If it is brittle it will flake. The pipes are immersed in the liquid and are left until they have assumed its temperature, and are then taken out to drain, and when the coating is hard they can be used. These tubes pass into the boxes on each side of the furnace water-tight. The cold water enters the lower half of the partitioned box through the four lower pipes, back by the six, and is discharged. In the hot condensers they answer perfectly. In the first condenser they lasted four years before they were taken out. In No. 3 they have been in two years with no sign of wear. They therefore cost next to nothing for wear, which comes exclusively from the careless coating of the pipes. Their installation is comparatively inexpensive. They serve to rapidly cool the hot gases. Before they were used the quicksilver did not condense much before the fourth or fifth condenser, and generally not until the seventh. Now it is collected in the first. This has resulted in the abandonment of the glass houses on the other side of the creek, and it is probable that they will be abandoned on this side. All of the condensers except No. 3, where the gases are cooled by the dryers, and the long iron pipes which lead to the condensing chambers, have two of these water backs, one in the descending and one in the ascending chamber.

From the last brick condenser the connection is made by a wooden flue 25 ft. long, with sixteen glass condensers arranged in somewhat irregular order but as the ground would allow of. Then 15 ft. of wood flue to a brick tower and 40 ft. of circular brick flue, 3 ft. 6 in. in diameter, and 183 ft. long, to the condensing system on the hill, with which No. 2 is also connected. The flue to the chimney is 130 ft. long. In 1886 a Guibal fan was put in here as the results of the experiments made with furnaces 3, 6, 7, 8, and 9, and now all the auxiliary fires are abandoned. They have not been taken away but are kept in place to be used in case of accident to the fan. On the office side of the creek the fans are run by steam, but on the other side they are run by electricity. The chimney with which this connects is 3 ft. 6 in. square, and 20 ft. high. It is 60 ft. above the top of the furnaces, and 100 ft. above the floor of the works. In 1888 the

wooden flue was extended 270 ft. up the hill to prevent the gases settling round the furnaces.

These shelf furnaces with their systems of condensation are the greatest improvement that has been made in the metallurgy of mercury in the United States. With the constant care and attention given to experimenting at New Almaden it is likely that they will be improved, but at present they seem to be the best and most economical furnaces yet invented.

The condensers produce both mercury and soot. The mercury is collected at once in receptacles and goes to the weighing and bottling rooms. Of these there are four, one for furnaces Nos. 6 and 8, next to No. 6, one for furnace No. 3, one for 7 and 9, and one for 1 and 2. These are situated in convenient positions, a little below the level of the floor of the works, so arranged that all the mercury from each furnace flows into a receptacle, to which a goose-neck is attached, from which, during the working of the furnaces, a continuous stream of mercury flows into cast-iron boilers about 4 ft. in diameter. They are not of any fixed size, being old vessels adapted to this use, some of them being evidently blubber trying out vats. As the weak acid water which flows with the mercury would attack them the stream from the furnaces is made to run into a wooden trough 1 ft. long 6 in. wide, and 10 in. deep. From the bottom of this there is a goose-neck for the discharge of the mercury, and the excess of acid water flows off by an overflow. In all the places where there is a goose-neck, or where the mercury drops, it is made to drop on a cloth. If the distance is too high, an india-rubber pipe is put upon the goose-neck. This is done in order to prevent the spattering of the mercury, and its breaking up into small globules. Usually the height of the boiler, which is about 30 in., is too great to admit of loss. From the boilers it is weighed in an iron pan scales, so arranged that the pan swings easily so as to tip into the funnel placed in the mouth of the flask; 76½ lb. is charged in each flask. The flask is then caught in a rectangular jawed vice. The stopper is screwed in tight with a wrench 5 ft. long by two men, and the flask is ready for shipment.

The flasks are purchased in the market second-hand. These are all examined and a very few rejected. The old corks are

thrown away, the holes beaten up and restoppered, and the flasks marked on the top with a large **A** with white paint. There is always some little mercury in them. When first received they are turned over a basin and tapped several times with a hammer, both to remove this mercury and the iron rusts. Five entire flasks were recovered from the mercury left in the old flasks in the course of fourteen months.

The soot contains considerable quantities of mercury mechanically mixed with it, a few salts of the metal, and a large quantity of carbon, and some tarry hydrocarbons, with some scrapings from the condenser walls. Most of the metal can be removed by purely mechanical means, but some of it cannot be separated in this way and is put back into the furnaces. From the first condensers and from the hill flues the soot is dry, but it is elsewhere damp. From the last condensers a black acid water runs out, with which there is some finely divided quicksilver. This is filtered through charcoal, the water flowing off is almost clear, it is settled and leaves a black mud, which is dried and treated in the furnaces. The mercury collected runs out through a goose-neck to the bottling room. The filters are taken up from time to time and are treated in the furnace.

The soot accumulates on the walls of the condensers, and when too heavy to hold itself falls off. When the condensers are cleaned while the furnaces are running, the soot on the bottom is worked with an iron hoe by the "soot men" as long as the mercury runs from it freely, and is then taken out. There is no danger in doing this, as the current of air is always inward; but only one condenser is opened at a time, for fear of interfering with the proper conduct of the draught. The walls are scraped down at the end of the campaign with a rubber 7 in. square, fastened to an iron plate 6 in. square, attached to a handle 12 ft. long. This brings down the soot without injuring the walls. When the furnace is out of blast and to be cleaned, the walls are washed down with a hose and the water allowed to settle in a tank. The condensers, while the furnace is running, are cleaned twice a month. The door of the condenser that has been cleaned is always closed and luted before the next one is opened.

The old method of treatment was to put the soot upon floors,

13 ft. by 10 ft., paved with cement and inclined 1 ft. in 10 ft., with a gutter 4 in. wide and 9 in. deep at the front, and inclining into a small depression used as a pot. The soot was vigorously stirred on the incline with wooden hoes until no more mercury could be extracted from it. It was then placed in cast-iron boilers, 42 in. in diameter, with inclined rims 5 in. wide, set three in a row in front of the cement floor and boiled with lye and vigorously stirred by hand. The mercury was collected and the residue was then dried and treated in the furnace. This method is now abandoned.

The soot now, as it is drawn from the furnaces, is placed on a floor of asphalte, 9 ft. wide and 20 ft. long, which is inclined towards the centre and one end, so as to be 14 in. deep at the lower end in the centre. It thus inclines from both sides and towards one end. As it comes from the furnace the soot is damp and often more or less of a mud. It is mixed with as much wood ashes as will dry it, which is usually 50 per cent. of the bulk, and any mercury that runs out is caught in the depression at the bottom of the incline. The ashes are used to dry up the moisture, so as to make a paste damp enough not to fly. It usually requires one pail of ashes to one of soot, which, however, does not use up all the ashes made in the works. It is then carried in iron pails to a machine which consists of a boiler 40 in. in diameter, with 5 in. flange, inclining on the bottom 2 in. in the 5 in., with a hinged valve 9 in. in diameter below for emptying it. There is a $1\frac{1}{2}$-in. hole in this bottom for the end of the revolving rod, which has $\frac{1}{16}$ in. play, for the mercury to run out. The vertical shaft has four arms shaped like a ploughshare, which are made to revolve in the soot forty times a minute for $1\frac{1}{2}$ hours. The charge is usually eight pails of soot and ashes. When first charged the mercury runs in a stream; 16 per cent. of the mercury comes out in the first half-hour, and in the end 90 per cent. of the mercury contained is recovered. No water is added, but the agitation is kept up until the mercury ceases to run. The bottom valve is then opened and the stem raised. The soot is pushed into the box below with a shovel and collected in iron pails. The last of it is taken up with a scoop. There is still a considerable quantity of

mercury, sometimes as high as 14 per cent., in it, which can be seen as very fine globules. It was formerly carried in the buckets to the furnaces and added in small quantities at a time. This furnace treatment is recognised as a bad method. The soot should be treated so that the quantity of mercury remaining in it will be too small to make it worth while to treat it so that it can be thrown away. It is now being fed automatically into an iron cube of about 3 ft. set on one of its angles, the interior of which is heated. The mercury volatilised passes up an iron pipe 4 in. in diameter, and then into a wooden box about 4 ft. cube which has a double top and bottom, which are connected by four rows of pipes 4 in. in diameter. The intervening space is filled with water. In the bottom there are four doors about 4 in. square; in the first of these mercury is distinctly seen, in the last none has as yet been found. The use of a hydraulic press has been suggested so that the mercury would be pressed out and the spent soot left in the shape of cakes, to be charged in the furnace, but the difficulty is its application. No process can be considered as successful which results in any part of the soot going back into the furnace.

Formerly not more than 4 to 5 per cent. of the quicksilver product of the works was from the soot. In 1888 they recovered 18 per cent. The "soot men" are paid a little more than the others. Salivation rarely occurs except as the result of carelessness. It generally comes from want of cleanliness or the use of tobacco, and, as there is no one to blame but themselves, no complaints are made.

The quantity of soot produced in the condensers varies very much, and depends on a number of circumstances which are more or less accidental, such as the quality of the ore treated and the kind of fuel used. The average for a considerable time from furnaces 3, 8, 7, and 9 is about 500 buckets per month, each bucket holding about half a cubic foot. This includes the wood ashes which are mixed with the soot to make it sufficiently dry to handle. The richness of the soot depends on the percentage of mercury contained in the ore, and also on the condition in which it is collected. From quite dry soot the mercury separates very easily and quickly, and what is taken from the condensers

will be poor. When the soot is damp, especially when it contains greasy hydrocarbons, the globules become coated on the outside, and do not coalesce, and such soot will be rich. There were collected in June, 1888, from five furnaces 563 buckets, yielding 16,712 lb. of mercury; in July, 1888, from four furnaces, 457 buckets, yielding 10,970 lb. of mercury; in August, 1888, from four furnaces, 543 buckets, yielding 9582 lb. of mercury; in September, 1888, from five furnaces, 512 buckets, yielding 12,024 lb. of mercury. The buckets of soot weigh according to their contents in mercury from 32 lb. to 60 lb. On the average they were 44 lb. The quality of mercury collected from the soot is about the same as that collected from the ore.

The following Table gives the operations of all the furnaces from May 1, 1887, to April 30, 1888:*

Furnace Number.	Days in Work.	Kind of Ore used.	Amount Treated in Ton of 2000 lb.		Quicksilver produced in Flasks of 76½ lb.	Quicksilver produced in Pounds.	Percentage of Yield.
			tons.	lb.			
1	236	Granzita	8,189				
		Granza	231	...	3,073	235,845	1.395
2	128	Granzita	2,113	1000			
		Granza	,261	1500	1,239	947,835	1.995
3	255	Tierras	9,111	...	3,078	235,467	1.292
8	200	,,	4,737	...	1,686	128,979	1.361
7	345	Granza	3,184	1600	7,062	540,243	8.481
9	157	,,	1,486	1000	3,050	233,325	7.848
6	23	,,	525	...	1,312	100,368	9.558
Total ...	1344		29,837	5100	20,500	2,422,062	31.930

This shows that the different classes of ore treated were:

	Tons.
Granza	5,689.0100
Granzita	10,302.1
Tierras	13,848
	29,839.1100

The quicksilver produced amounted to 784,250 tons. From July, 1880, to April 30, 1888, this company has produced 880,718 flasks.

* Report of the Quicksilver Mining Company, 1888, p. 47.

The following Table gives the details of cost for 1888:

		$
Staff and office expenses ...		0.806
Excavations ...	Mine expenses.	10.476
Strengthening ...		.996
Drainage		.252
Tramming		.157
General repairs ...		1.84
Supplies		5.75
Distillation and flasking ...		3.499
Workshops...		.452
Transportation		.737
		$24.965

The following Table gives the cost per flask from 1884 to 1888:

——	1884.	1885.	1886.	1887.	1888.
Number of flasks produced	20,000	21,400	18,000	19,550	20,500
	$	$	$	$	$
Mine expenses . . .	14.006	15.565	18.216	16.301	14.517*
,, supplies . . .	4.693	5.755	5.522	4.618	3.776
Expenses at works . .	2.648	2.549	3.150	2.840	2.633
Supplies at works . .	2.290	2.515	2.450	2.226	1.902
Taxes	1.524	1.38	1.48	1.375	1.380
Total cost . . .	25.161	27.764	30.818	27.36	24.208
Less rents	1.16	1.003	1.678	1.51	.934
	24.001	26.761	29.14	25.85	23.274
Credit to ore account . .	191	655	...	1.47	1.462
	24.192	27.416	29.14	27.32	24.736

* Includes $4.661 per flask for prospecting.

In the year 1888, 10,491 tons were mined, of which 79,611 were waste rock; 6277 tons of ore were obtained from the surface, making 31,157 tons of ore produced. There were 627 tons in the sheds at the commencement of the year, making in all 31,785 tons, of which 29,839 tons were worked. These ores yielded 2.627 per cent, and produced 20,500 flasks of quicksilver.

Fuel has become so dear at New Almaden that crude petroleum is now used for making steam for pumping, and it is believed that, on account of the growing scarcity of wood, gas will soon have to be used in the furnaces.

The Table on the next page, compiled by J. B. Randol, gives the production of quicksilver at New Almaden for thirty-five years and three months, from July, 1850, to 31 December, 1887.

It will be seen from this Table that the greatest number of flasks ever produced in any one year was in 1865, when 47,194 flasks were produced from 31,948,400 lb. of ore, yielding 11.3 per cent. That the greatest number of tons of ore treated in any one year was in 1886, when 81,398,690 lb. were treated, yielding only 18,000 flasks, or a yield of 1.69 per cent. This being the lowest average ores ever treated. The smallest number of flasks produced previous to 1858 was 18,035 from 28.50 per cent. ore in 1853. Within the last twenty years the highest yield was 29,000 flasks in 1883 from 77,162,500 lb. of 2.87 per cent. ore, and the lowest 9084 flasks in 1874 from 23,454,000 lb. of 2.96 per cent. ore. The highest percentage of mercury in the ores was in 1850, when it was 36.74 per cent., since which it has steadily fallen with only occasional fluctuations. Within the last twenty years the lowest percentage was 1.69 per cent. in 1886, and the highest 6.63 per cent. in 1872. It is safe to say that without the continual and intelligent attention which has been given both to the mining and treatment, the works must long ago have closed, like many others. They are more intelligently worked to-day than in the times of their greatest prosperity, though it has been many times announced that the mine had given out.

Livermore Furnace.

A fine ore furnace designed and constructed by Mr. C. E. Livermore, of the Redington Mine, has worked successfully there and elsewhere. It is shown in the plan as constructed at the Redington Works in Fig. 137, and in section in Fig. 138. It consists of a series of channels 30 ft. long, or longer if desirable, inclined at an angle of 50 deg., built upon the side of a hill, with a charging hopper at the top and a fireplace at the bottom. These channels B, Fig. 138, are built of fire-brick, and in order to compel the ore to remain a certain time on the hearth, firebricks 2½ in. by 4 in. I, cut roof-shaped, are placed so as to form shelves or riffles against which the ore falls, and prevents its too rapid movement downward.

Dates.	Class and Quantity of Ore.			Total Pounds.	Flasks from Furnaces.	Flasks from Washings.	Flasks, Total.	Average Amount per Month. Flasks.	Percntge. Yield of Quicksilvr.	Number of Months.
	Grueso, Pounds.	Granza, Pounds.	Tierras, Pounds.							
July 1850 to June 1851				4,970,717	23,875		23,875	1,989½	36.74	12
July 1851 to June 1852				4,643,290	19,921		19,921	1,660	32.82	12
July 1852 to June 1853				4,339,520	18,035		18,035	1,503	28.50	12
July 1853 to June 1854				7,448,000	26,325		26,325	2,193¾	27.03	12
July 1854 to June 1855				9,109,300	31,860		31,860	2,655	26.75	12
July 1855 to June 1856				10,355,200	28,083		28,083	2,340½	20.74	12
July 1856 to June 1857				10,299,900	26,002		26,002	2,167	19.31	12
July 1857 to June 1858				10,997,170	29,347		29,347	2,445½	20 41	12
July 1858 to Oct. 1858				3,873,085	10,588		10,588	2,647	20.91	4
Nov. 1858 to Jan. 1861	Closed by injunction.									
Feb. 1861 to Jan. 1862				13,323,200	32,402	2,363	34,765	2,897	19.96	12
Feb. 1862 to Jan. 1863				15,281,400	39,162	1,129	40,291	3,366	20.22	12
Feb. 1863 to Aug. 1863				7,172,660	17,316	2,248	19,564	2,796	20.86	7
Sep. 1863 to Oct. 1863				2,346,000	4,820	700	5,520	2,760	18.00	2
Nov. 1863 to Dec. 1863	54,800	1,586,500	718,000	2,359,300	4,040	407	4,447	2,223½	18.85	2
Jan. 1864 to Dec. 1864	1,259,400	18,730,300	3,287,900	23,277,600	42,176	313	42,489	3,540¾	13.96	12
Jan. 1865 to Dec. 1865	2,288,900	25,749,000	3,910,500	31,948,400	47,078	116	47,194	3,933	11.30	12
Jan. 1866 to Dec. 1866	1,506,000	19,939,100	5,440,200	26,885,300	34,726	424	35,150	2,929	10.00	12
Jan. 1867 to Dec. 1867	731,500	15,689,288	9,603,145	26,023,933	23,990	471	24,461	2,038½	7.19	12
Jan. 1868 to Dec. 1868	2,274,208	14,566,600	12,564,722	29,405,530	25,877	51	25,928	2,135¾	6.66	12
Jan. 1869 to Dec. 1869	150,000	11,942,175	13,366,000	25,458,175	16,898		16,898	1,408	5.07	12
Jan. 1870 to Dec. 1870	30,000	12,531,900	8,535,800	21,097,700	14,423		14,423	1,202	5.23	12
Jan. 1871 to Dec. 1871		13,661,700	8,373,000	22,034,700	18,563	5	18,568	1,547½	6.44	12
Jan. 1872 to Dec. 1872	142,000	12,777,000	8,497,600	21,416,600	18,391	183	18,574	1,548	6.63	12
Jan. 1873 to Dec. 1873		8,492,375	8,828,000	17,320 375	11,042		11,042	920	4.87	12
Jan. 1874 to Dec. 1874		11,294,000	12,160,000	23,454,000	8,867	217	9,084	757	2.96	12
Jan. 1875 to Dec. 1875		12,236,000	18,870,200	31,106,200	13,541	107	13,648	1,137½	3.35	12
Jan. 1876 to Dec. 1876		14,784.550	18,532,400	33,316,950	20,549		20,549	1,712½	4.71	12
Jan. 1877 to Dec. 1877		13,987,700	23,243,600	37,231,300	23,996		23,996	1,999¾	4 93	12
Jan. 1878 to Dec. 1878		14,612,154	22,330,071	36,942,225	15,852		15,852	1,321	3.28	12
Jan. 1879 to Dec. 1879		16,032,085	39,033,050	55,065,135	20,514		20,514	1,709½	2.85	12
Jan. 1880 to Dec. 1880		15,267,650	46,067,200	61,354 850	23,465		23,465	1,955¼	2.92	12
Jan. 1881 to Dec. 1881		14,430 510	49,710,625	64,141,135	26,060		26,060	2,171¾	3.10	12
Jan. 1882 to Dec. 1882		19,734,900	52,412,300	72,147,200	28,070		28,070	2,339	2.97	12
Jan. 1883 to Dec. 1883		21,227,500	55,935,000	77,162,500	29,000		29,000	2,416¾	2.57	12
Jan. 1884 to Dec. 1884		16,410,000	62,841,000	79.251,000	20,000		20,000	1,666⅔	1.93	12
Jan. 1885 to Dec. 1885		17,644,300	61,425,000	79,069,300	21,400		21,400	1,783	2.07	12
Jan. 1886 to Dec. 1886		14,140,690	67,258,000	81,398,690	18,000		18,000	1,500	1.69	12
Jan. 1887 to Dec. 1887		12,648,300	51,503,000	64,151,300	20,000		20,000	1,666⅔	2.38	12
Totals and averages	8,436,808	370,116,277	664,476,318	1,147,688,840	854,054	8,734	862,788	2,089	5 75	423

Product of Enriqueta from 1860 to 1868, 10,571 flasks. Total product of all the mines on the Company's property, 873,359 flasks of 76½ lb. each, or 66,811,963½ lb.

The firebricks are set 12 in. apart and in the first furnaces were 2 in. from the hearth at the fireplace, the distance gradually increasing until it is 4 in. at the hopper. They were kept low for fear that the ore would not be perfectly roasted if they were higher. They were finally raised to 5½ in. and have been retained at that height as the ore is always perfectly roasted. In the roof there are projections H, of square brick 4 in. in depth, to deflect the flame against the ore, instead of allowing it to follow the roof as it naturally would. These flame deflectors are placed in the intervals between the ore shelves I. The exact arrangement of these ore channels is seen in Fig. 138. The first furnace constructed had eleven of these ore channels, but as many as twenty have been built in one furnace. They are 12 in. deep and 6¾ in. wide. The only limit to their number will be the double length of a fireplace. After the ore has traversed the hearth, the length of which depends upon the length of the incline of the hill, it falls through a flue made on the hearth side of the firebridge into a cooling chamber L, Fig. 136, where it cools before being drawn out into the iron discharge car. This chamber connects by flues with the condensers, so that the vapours given off by the ore will be carried to them. In the bottom of the furnace there are peep-holes K, Figs. 138 and 139, for each ore channel, so that its operation at that point can be watched and the condition of the ore distinctly seen. For the large furnaces the fireplaces are double, for the small ones they are not so. For the double furnace the fireplace is 26 in. wide, 6 ft. long, and 24 in. high, from the grate to the arch. The firebridge is 20 in. high above the grate and 18 in. from the roof. Between the bridge and the roof there is a partition to scatter the flame. In the first furnaces constructed the floor of the channel in its upper part was made thin in the large furnace, and an under fire arranged in case it might be necessary to use it as shown in Fig. 138. This under fire seems a very doubtful improvement. It is not used in the small furnaces, and after trial was finally abandoned as it only wasted fuel. In order to make the roof of the furnace as non-conducting as possible it is covered with ashes.

About half way up the incline of the furnace there is a dust

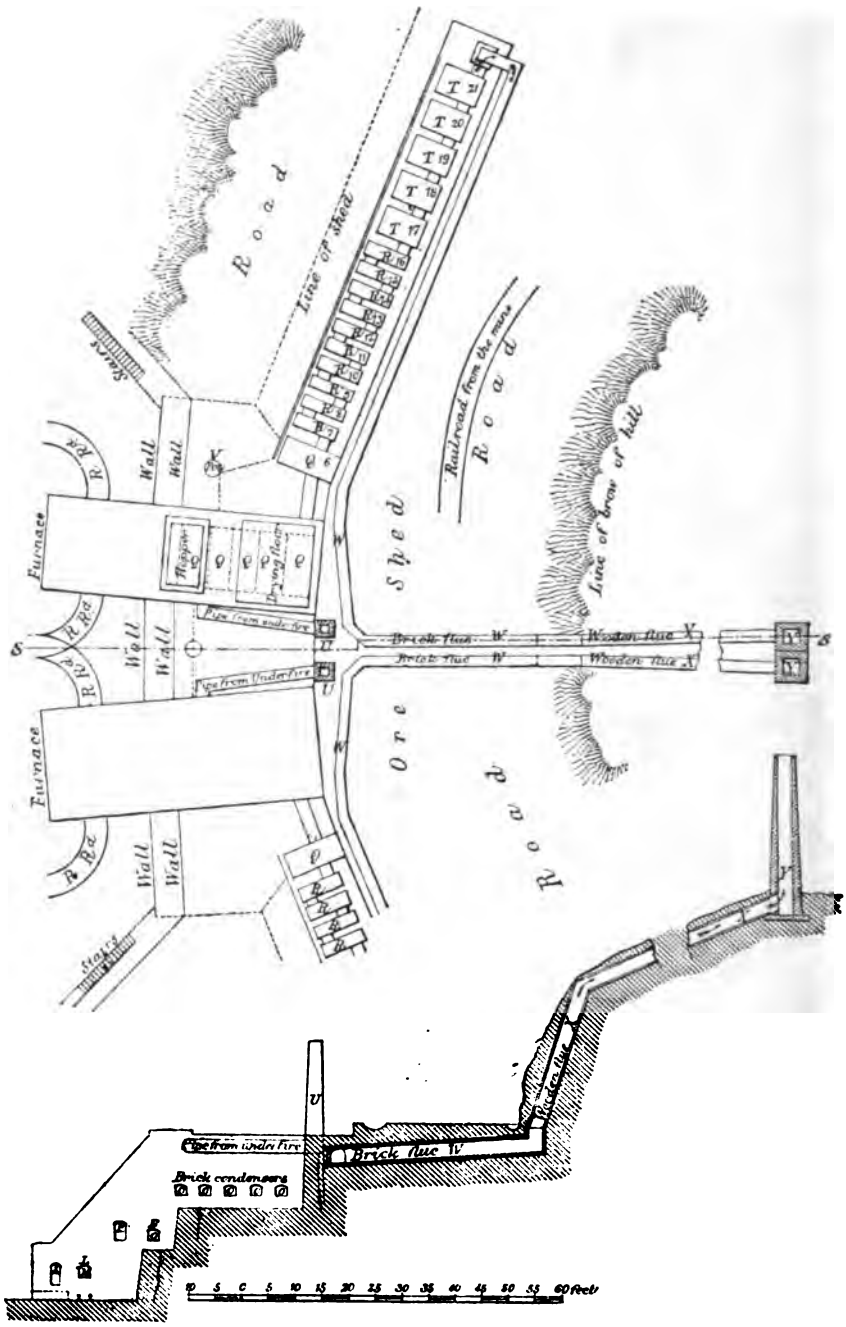
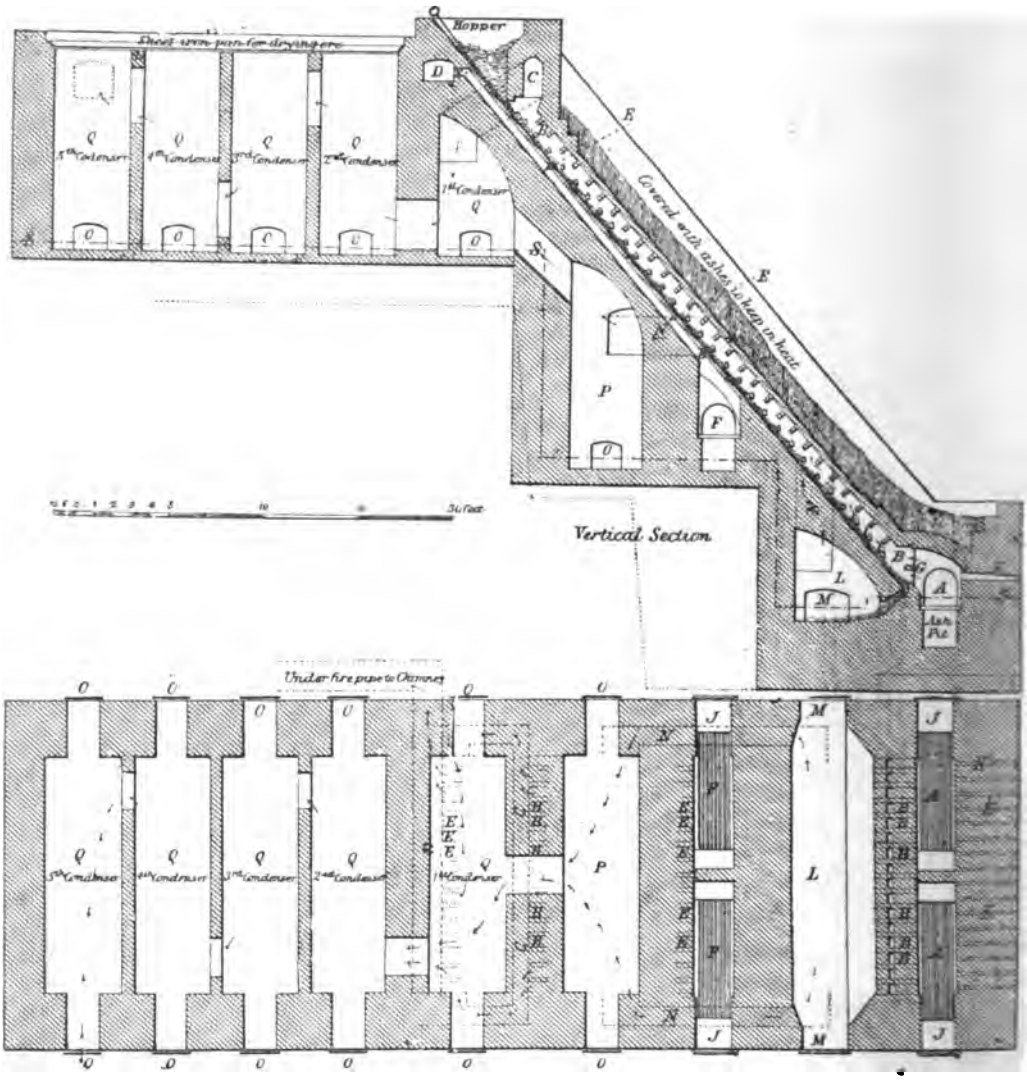


FIG. 137. PLAN AND SECTION OF THE LIVERMORE FURNACES.



To face page 891. FIG. 138. SECTIONS OF THE LIVERMORE FURNACE.

chamber P, Fig. 138, to catch all the fine ore which may be carried off from the cooling chamber, and connected with it a series of five brick condensers Q, 4 ft. wide, 10 ft. 9 in. high, and 15 ft. long, connecting one with the other. These condensers are covered with iron pans, like the old Idria condensers, which are used for drying the ore. Following these are ten iron condensers R, Fig. 137, of the same size as those for the Knox furnace, and following them are five wooden condensers, shown in Fig. 140, 8 ft. long and 6 ft. high at one end and 5 ft. at the other, and 6 ft. wide, made of red wood timber, and braced on the outside by timbers 4 in. by 6 in., against braces 4 in. by 4 in. on the inside. Of course no nails can be used here on

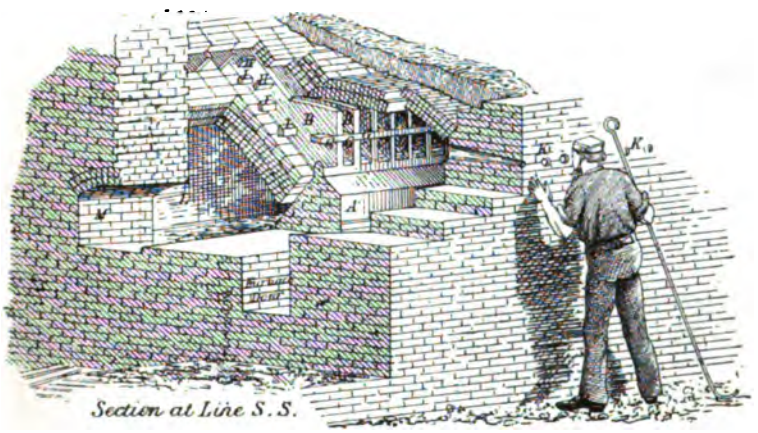


FIG. 139. PERSPECTIVE VIEW AND SECTION OF THE LIVERMORE FURNACE.

account of the sulphurous fumes; from here the fumes discharge into a long brick flue W, Fig. 137, leading to a wooden one X, which ends in a chimney Y.

The ore from the mine was discharged over the screens formerly used for the modified Idria furnace, except that the lower screen was of wire with 1¼-in. meshes, instead of ¾ in., as formerly. It was made into three sizes; the large pieces were hand-picked and broken, and with the second size, which passed through the first screen, were charged into the Knox furnace. The third size, or the fine ore, went to the fine ore furnace as it came from the screens. Since the introduction of this furnace at the Redington Mine all of the ore is crushed and screened to

reduce it to fine material; their experience having demonstrated that this is the most economical method of treating the ore. The ore from the mine is dumped on a platform paved with stone, which has an opening in the centre which is covered with heavy bars of iron placed from 6 in. to 8 in. apart. The small pieces pass

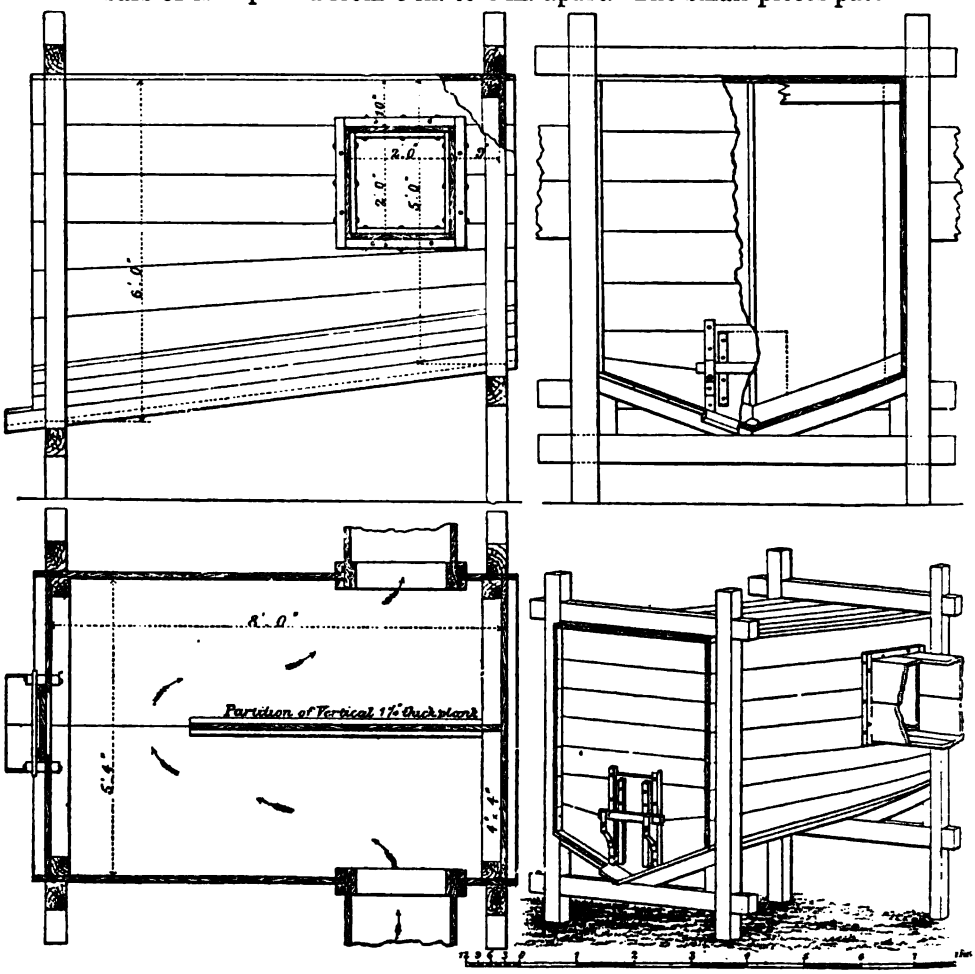


FIG. 140. WOODEN CONDENSER FOR THE LIVERMORE FURNACE.

through, and those which do not are broken by hand until they pass the bars. From here the ore falls on to another screen with meshes 1½ in. square. All that does not pass this screen falls into a hopper which feeds an ore crusher, which is run by a small upright engine. Under the centre is a small shaking screen.

All pieces which do not pass this screen fall into a small bin and go back to the crusher. The ore which has passed the screens falls into a large bin, from which it is carried in cars drawn by mules to the furnaces.

The ore is charged by a hopper, which is 2 in. by 2 in. at the bottom, and which is kept from being stopped by a long iron rod, with which the bottom of the flue is constantly sounded. A furnace, with ten to twelve channels, will treat 17½ to 21 tons of ore in twenty-four hours, or about 1¾ tons per channel. The large furnaces will reduce 1 ton per hour or more according to the number of ore channels. At the Redington Mine each channel will treat 1¾ tons in twenty-four hours, each cubic foot of average ore weighing 115 lb. The ore here is considerably lighter than that used in the Knox furnace, since it is previously dried. The amount of fuel required for a 10 and 12-channel furnace is 1½ to 2 cords of wood per twenty-four hours. After treatment the ore, which when charged weighed 115 lb., will not weigh over 55 lb. Six men in two shifts of twelve hours each are required per furnace. An 11-channel fine ore furnace will, if pushed, burn one car-load per hour, each car-load containing 15 cubic feet.

A 12-channel furnace costs, completed, including the buildings, about $8000. One furnace capable of treating 15 to 16 tons of ore in twenty-four hours has been put up at the Sunderland Mine. They use wrought-iron condensers, 18 in. in diameter, and produce the draught by means of a steam jet in the wooden flue.

The 20-channel furnaces are arranged in pairs, and are fitted with

7 brick condensers,
14 iron condensers,
7 wooden condensers,
150 ft. of brick flue,
85 ft. of wooden flue,*
30 ft. of wooden chimney.

With such a pair of furnaces 60 tons† or more of ore are treated every twenty-four hours.

* These flues are too short; it would be much better to have them longer.
† A ton of ore at the Redington Mine measures 18 cubic feet.

There was formerly at the Redington Mine one 9-channel furnace, which was afterwards made into an 11-channel, and one 11-channel, both of which were made out of the condensers of the modified Idria furnace, and have but one fire; and one 16 and one 20-channel furnace, both of which have under-fires.

At the Redington Mine these furnaces have given the best results obtained there with fine ore. Its cost both of construction, running, and repair, have been found to be less. The furnace is more easily run, and can be started and stopped more readily, and is easier to clean when clogged, which occurs in all kinds of fine ore furnaces when the ore is wet, than any ever used there. It also requires less fuel to run it than a shaft furnace. On account of the angle at which the furnace is placed it can be run without a blower or exhaust. The quantity of air can be so regulated as always to be a maximum, so as to burn the greatest quantity of sulphur, and it runs a long time without needing repairs of any kind.

As is seen, these furnaces do away with the necessity of making adobes, and a considerable amount of the labour of picking the ore. They worked for several years very successfully, and were a real advance in the progress of the metallurgy of mercury.

With regard to these systems of working, it may be said that the apparent advantage of the retort system is its economy of installation; it does not appear that there is any gain in yield, and while the expenses for fuel and labour are necessarily a maximum, the amount which can be treated at any given time must be considered as a minimum. The cost of repair is also a maximum, as the iron of the retort is rapidly attacked by the sulphur of the ore, so that a large number of retorts must always be kept on hand. The danger of accident is also at a maximum, and there seems to be no good reason why such furnaces should not go out of use altogether. It is greatly to be regretted that the books which cite experiments that were made nearly one hundred years ago, should still continue to unequivocally praise this method, which seems only applicable for the treatment of ores when a new mine is being opened, in order to get some

return before the permanent furnaces are built. It does not seem even then to be always very desirable.

The modified Idria furnace, though a great improvement on its original, is a cumbersome structure, costly both to build and to use. With the exception of a single furnace still remaining at the New Almaden Mine, it has been abandoned. It involved such an outlay of capital for its construction, while it did not do its work as well as the modern furnaces, that it will hardly find advocates among modern metallurgists.

Of the continuous shaft furnaces, the Knox held its ground against a host of competitors for a number of years. It had for a long time no real competitor, but the adverse decisions of the Courts as to the validity of its patents, and the constant improvements in the furnaces, rendered necessary by the low price of mercury, and suggested by experience in other furnaces, have driven it out of use, so that it was at the last used almost exclusively at the Redington Works, as almost all the other works which used it have been for a long time idle. For fine ore the Livermore furnace has not only been successful, but was at one time almost without a competitor. Of all the furnaces now working on account of the constant and successful experiments made there, the best are undoubtedly those now used at New Almaden.

There is evidently a limit beyond which the speed of the blowers, which are so characteristic of California practice, cannot be increased. For perfect condensation a very large surface is necessary, but a certain amount of time is quite as indispensable to cool the volatilised material in order to have it condense. If, therefore, even though the condensing surface be very large, the velocity is at a maximum, it is evident that there will be a loss of mercury which will be exhausted out of the condensers by the rapidity of the current. The proper speed of the blower will vary at different times with the obstruction caused to the passage of the air by the size of the ore, and the kind of condensers used. It is evident that in a given furnace the ore may be charged so fine that it will pack in such a way that no speed of the blower will draw the furnace gases through it, or, on the other hand, it may be so large that only a few

revolutions per minute will be required. There is evidently a mean between these two extremes, which can only be determined by the person in charge of the furnace, and it is evidently this judgment and skill, which may or may not have been acquired by long experience, which makes an inferior process give a good result in one place, and the want of it which akes a good process fail in another. Such examples occur, not only in the metallurgy of mercury, but throughout the whole range of metallurgical industry.

The objection to masonry condensers is generally the penetrating power of the mercury, which is so great that it not only impregnates the vertical stone or brick walls of the chambers, but infiltrates into the ground to such an extent that in taking up the foundations of some old furnaces at New Almaden in 1863, the mercury was found to have penetrated into the foundations and into the clay beneath it to the bedrock, a depth of 25 ft. or 30 ft. Over 2000 flasks were recovered in a single year from the foundations of two furnaces. This loss is now prevented by building the condensing chambers with double arches into which plates of iron are built so as to cut off the flow of mercury, and turn it towards an iron basin into which it may be seen to flow in intermittent streams every few minutes. In the year 1874 a large shaft 30 ft. deep was dug beside one of the furnaces, and at this depth it was still found profitable to wash the dirt in order to collect the mercury from it. In the same year sufficient mercury was obtained from a brick ore chamber and three condensing chambers of a furnace, and from the dirt around the structure, to pay for a new shaft furnace and condensers.

For the construction of condensers, bricks, though porous, are much preferable to, and less porous than, the sandstone used at the Redington Mine. Whenever repairs to the condensers at these works have been made with bricks they have all stood, while the sandstone next to the brick has been worn away. It is evident that what is required is some substance which is impervious to mercury. Brick is not so. Every brick or piece of sandstone taken out of the condenser must be considered and treated as a very rich ore, for whenever it is broken, myriads of globules

of mercury are seen scattered through it. New brick, when dry, may be rendered completely impervious to mercury by coating it with asphalte. This cannot be made to stick after the brick has been exposed for some time to the acid fumes, and salts have commenced to form in its interior. Iron is better than naked brick, in that it does not absorb mercury, but it is rapidly corroded by the cold weak acid vapours. At the Sulphur Bank Mine the amount of iron eaten away in 1875 by these waters amounted to three tons per month. Wood is much cheaper than brick, but it cannot be used near the furnace on account of the heat. At a certain distance from it, it is more or less rapidly acted upon by the acid vapours. It is kept from shrinking by the condensed moisture, but requires watching. If such condensers were used where the mercury condenses in any considerable quantity, there would be danger of a small loss from absorption in the pores of the wood; or, if they were not carefully watched, from cracks. When they require to be renewed, all the wood should be burned in the furnace, though the mercury so collected would not be very large in amount.

It seemed at one time perfectly practicable to make such a combination of wood and glass that it would stand at a certain distance from the furnace; or by placing brick condensers near the furnace and cutting them up into small structures so that they should not retain the heat; or by substituting iron condensers, with water cooling arrangements, in connection with wooden condensers, to make a condensing apparatus very nearly perfect. By using, for instance, the Knox condensers near the furnace, with a wood or a wood and glass arrangement at the point where there is no danger from heat, a considerable amount of expense might be saved; such condensers, Fig. 140, have been used at the Redington Mine made of sound red wood 1¼ in. thick, nearly double the size, but of the same shape as the iron ones, that are found to last equally well, and are far cheaper. In any case a few large compartments are not so effective as the same volume distributed among a number of small ones. To effect a perfect condensation a cooled surface is more necessary than volume,* and more espe-

* Treatment of Flue Dust at Ems. Trans. Am. Inst. Min. Eng., vol. xi., p. 379.

cially sudden and frequent changes of the direction of the current of the volatilised mercury. With a furnace having only a natural draught this is impracticable; but with a furnace working with an exhaust fan, the theoretical limit of obstruction is the power of the engine which produces the draught. There is, however, a practical limit of obstruction which cannot be overstepped. Nothing has, as yet, equalled the water back condensers at New Almaden. Their construction is simple and inexpensive. They cost nothing for repairs, when they have been properly coated, and up to the present time they give the most satisfactory results, although they are placed in the most unfavourable positions.

There seems to be a decided advantage in using artificial fuel where it can be had. European experience has pronounced in favour of it. At New Almaden experiments were made in 1873 and 1874 with coke and charcoal, and a mixture of both, which were very satisfactory. Later experiments have shown coke to be the best. It would even seem advantageous under certain circumstances to mix a certain amount of fuel with the ore, especially when large amounts of fine ore not made into adobes must be charged. It is evident that the fuel which contains the least amount of moisture, if it is not friable, will prevent the formation of a certain quantity of soot, and as artificial generally contains less moisture than natural fuel, there will probably be less acid condensed. There are places, however, where the cost of making artificial fuel would more than counterbalance the advantage gained.

It is very difficult to get any idea of the loss in these furnaces since assays are not usually made. The idea of sampling the ore does not seem formerly to have entered into the calculations of many of the most intelligent managers, and this is natural enough. The enormous profits made when mercury ruled at from $1.50 to $1.75 per pound, rendered it entirely unnecessary to look after such small matters of detail. It is necessary, however, now, and will become more so in the future, for the hosts of patentees of the "best and cheapest quicksilver furnaces," with mercury about $0.40 the pound, to substantiate their claims on something better than assumption or newspaper advertisements. The losses at Idria in Austria are reported to have been, for 1870, 6.9 per cent.; for

1871, 9 per cent.; for 1872, 10.6 per cent. There seems to be a doubt about their method of sampling. At New Almaden, after taking a sample from each car-load for thirty-nine days, they report their loss to be 7.29 per cent. The difficulty of sampling ores of mercury, and the probability of getting too low rather than too high an average, makes the loss appear much lower than it really is.

According to the best California authorities, the loss in the best constructed furnaces, as near as can be approximated, is not less than 15 to 20 per cent.; and in many of the works the loss will probably amount to double that. In the best works it would probably cost more to save the 15 or 20 per cent. lost than it is worth; and the works which are not well managed probably could not save it at all. There are undoubtedly many works which have failed financially, who could have declared a dividend on what they might have saved. The reign of economy and of true metallurgical work must come with the era of low prices.

Every system which has been used, and some even after failures, have their supporters. When the date of the discovery of these deposits in California is taken into consideration, it must be confessed that very rapid strides have been made in the metallurgy of quicksilver there. There is, however, much still remaining to be done. So little is known of what is done, and the prices of the metal have been subject to such frequent fluctuations, that except in the case of the wealthy companies, there has been but little encouragement to experiment. When the price of quicksilver in San Francisco ranged from $1.50 to $1.75, every place where any trace of mercury could be found, became a mine, and reducing works, more or less extensive, were erected. With the fall in price, most of these companies disappeared entirely. In former years the price has fallen as low as $0.35 per pound, and with the price of $34.45 to $48 per flask, as it has ruled from 1880 to 1888, few of such small works can compete.

The fluctuations in price of the metal are excellently illustrated by the following Table,* which gives the market price per pound for the year 1876. How frequent they were, will be seen from the annexed extract from the account sales.

* For the following statistical tables I am indebted to Mr. J. B. Randol of New Almaden.

	Cents.		Cents.
Jan. 3	65	Aug. 7	44
,, 13	62½	,, 15	45
,, 20	60	,, 26	44
Feb. 3	62½	Sept. 1	45
,, 5	65	,, 11	44
,, 28	59	Oct. 1	45
May 1	58	,, 10	44
,, 20	54	,, 14	45
,, 25	52½	,, 30	50
June 5	51½	Nov. 3	52½
,, 14	50	,, 6	55
July 15	45	Dec. 12	50
,, 26	43	,, 31	50
Aug. 4	45		

Opening	65 cents	per pound	equal to	$49.725 per flask.
Closing	50 ,,	,,	,,	38.25 ,,
Range	15 ,,	,,	,,	11.475 ,,

Under the circumstances shown by these variations in price the small companies are obliged to wait to benefit by the experiments and experience of the larger companies, if they are not entirely ruined by the fluctuations before they have the opportunity.

The following Table gives the price per flask of 76½ lb. avoirdupois corresponding to the price per pound and *vice versâ*.*

Per Flask.	Equivalent in Cents per Pound.	Per Flask.	Equivalent in Cents per Pound.	Per Flask.	Equivalent in Cents per Pound.	Per Flask.	Equivalent in Cents per Pound.
dols.		dols.		dols		dols	
26.00	33.987	29.00	37.911	32.00	41.835	35.00	45.759
26.25	34.314	29.25	38.238	32.25	42.162	35.25	46.078
26.50	34.641	29.50	38.565	32.50	42.489	35.50	46.405
26.75	34.968	29.75	38.892	32.75	42.816	35.75	46.732
27.00	35.295	30.00	39.219	33.00	43.143	36.00	47.059
27.25	35.622	30.25	39.546	33.25	43.470	36.25	47.386
27.50	35.950	30.50	39.873	33.50	43.797	36.50	47.713
27.75	36.276	30.75	40.200	33.75	44.124	36.75	48.040
28.00	36.603	31.00	40.527	34.00	44.451	37.00	48.366
28.25	36.930	31.25	40.854	34.25	44.778	37.25	48.693
28.50	37.257	31.50	41.181	34.50	45.105	37.50	49.020
28.75	37.584	31.75	41.508	34.75	45.432	37.75	49.347

The following Table shows the production in flasks of the only mines which were working during the year 1889:

* Spanish and Austrian quicksilver flasks contain 75 lb. Spanish, equal to only 76.03 lb. avoirdupois.

TY-NINE YEARS.

A.

Kentuck.	Various Mines.	Total Yearly Production of California Mines.	Exports from San Francisco by Sea and Land.	Price in San Francisco per Flask. Highest.	Price in San Francisco per Flask. Lowest.	Production of the Idria Mine. (Austria).	Production of the Almaden Mine. (Spain). In Periods of Five Years.	Price in London per Flask. Highest. £ s. d.	Price in London per Flask. Lowest. £ s. d.	Year.
lks.	flasks	flasks	flasks	dols.	dols.	flasks	flasks			
*	...	7,723	6,467	114.75	84.15	4,100		15 0 0	13 2 6	1850
...	...	27,779	10,791	76.50	57.35	4,092		13 15 0	12 5 0	1851
...	4,099	20,000	21,458	61.20	55.45	4,085	101,517	11 10 0	9 7 6	1852
...	...	22,284	18,800	55.45	55.45	4,409		8 15 0	8 2 6	1853
...	...	30,004	20,963	55.45	55.45	4,060		7 15 0	7 5 0	1854
...	3,858	33,000	27,165	55.45	51.65	4,446		6 17 0	6 10 0	1855
...	2,862	30,004	23,740	51.65	51.65	5,935		6 10 0	6 10 0	1856
...	...	28,204	27,262	53.55	45.90	9,189	110,058	6 10 0	6 10 0	1857
...	5,239	31,000	24,412	49.75	45.90	4,977		7 10 0	7 5 0	1858
...	11,706	13,000	3,399	76.50	49.75	8,239		7 5 0	7 0 0	1859
...	2,939	10,000	9,488	57.35	49.75	4,821		7 0 0	7 0 0	1860
...	571	35,000	35,995	49.75	34.45	6,493		7 0 0	7 0 0	1861
...	1,885	42,000	33,747	38.25	34.45	4,712	122,117	7 0 0	7 0 0	1862
...	6,876	40,531	26,014	45.90	38.25	5,878		7 0 0	7 0 0	1863
...	2,286	47,489	36,927	45.90	45.90	7,263		9 0 0	7 10 0	1864
...	2,261	53,000	42,469	45.90	45.90	4,908		8 0 0	7 17 6	1865
...	2,621	46,550	30,287	57.35	45.90	5,327		8 0 0	6 17 6	1866
...	3,184	47,000	28,853	45.90	45.90	7,532	153,224	7 0 0	6 16 0	1867
...	112	47,728	44,506	45.90	45.90	8,253		6 17 0	6 16 0	1868
...	...	33,811	24,415	45.90	45.90	9,179		6 17 0	6 16 0	1869
...	...	30,077	14,240	68.85	45.90	10,745		10 0 0	6 16 0	1870
...	840	31,686	16,339	68.85	57.35	10,904		12 0 0	9 0 0	1871
...	...	31,621	16,780	66.95	65.00	11,116	165,608	13 0 0	10 0 0	1872
...	3,276	27,642	11,164	91.80	68.85	10,939		20 0 0	12 10 0	1873
...	...	27,756	11,750	118.55	91.80	10,789		26 0 0	19 0 0	1874
...	3,591	50,250	37,829	118.55	49.75	10,717		24 0 0	9 17 6	1875
54	1,161	75,074	49,046	53.55	34.45	10,794		12 0 0	7 17 6	1876
...	250	79,396	52,695	44.00	30.60	11,020	208,200	9 10 0	7 2 6	1877
...	...	63,880	41,877	35.95	29.85	10,403		7 5 0	6 7 6	1878
...	84	73,684	62,845	34.45	25.25	11,153		8 15 0	5 17 6	1879
...	...	59,926	46,294	34.45	27.55	12,356	41,640*	7 15 0	6 7 6	1880
...	376	60,851	45,799	31.75	27.90	11,333	50,353	7 0 0	6 2 6	1881
...	241	52,732	40,417	29.10	27.35	11,663	46,591	6 5 0	5 15 0	1882
...	101	46,725	37,867	28.50	26.00	13,152	46,143	5 17 6	5 5 0	1883
...	7	31,913	21,901	35.00	26.00	13,967	43,099	...	...	1884
...	392	32,073	25.730	32.00	28.50	13,503	46,739	...	...	1885
...	786	29,981	16.301	39.00	32.00	14,496	51,199	...	...	1886
...	455	33,760	22.119	48.00	36.50	14,676	53,276	11 0 0	6 10 0	1887
...	992	33,250	...	48.00	37.00	14,962	51,872	...	...	1888
54	63,051	1,518,380		118.55	25.05	347,586	1,291,636	...	...	Totals

voirdupois.

* Yearly.

ish = 34,507 kilogrammes = 76.07 avoirdupois.

New Almaden	13,000
Napa consolidated	4,500
Great Western	550
Sulphur Bank	2,150
New Idria	1,000
Great Eastern	1,350
Redington	800
Bradford	1,700
Various mines	500
Total	25,550

The lowest price per flask during the year was $40, the highest $50, the average price was $45. The table below gives the value of the quicksilver produced in California for the last eleven years at its average price.

	$		$
1879	2,200,000	1885	970,000
1880	1,860,000	1886	1,060,000
1881	1,810,000	1887	1,425,000
1882	1,500,000	1888	1,415,000
1883	1,275,000	1889	1,154,000
1884	975,000		

The folded Table opposite shows that while Almaden in Spain has been in the last thirty-nine years the greatest producer, having put on the market 1,291,636 flasks, New Almaden has produced in the same time 890,859 flasks, or only about one-third less, and that it has produced more than twice as much as Idria in Austria which has produced in the same period 347,586 flasks, and that during the same period, all the mines of California together have produced 1,518,380 flasks, 226,741 flasks more than Almaden in Spain, and only 120,842 flasks less than both Idria and Almaden in Spain put together. While, as the Table shows, the life of many of the mines in California has been short, many of them lasting only from three to five years, New Almaden has produced continuously, and in the years of highest production has turned out from one-third to more than one-half of the total production of the State. The fluctuations in price have also been very remarkable, beginning in 1850 at $114.75 per flask, it gradually fell to $75.25 in 1879, having been as high as $118.55 in 1874-5. Since 1880 it has been gradually rising until in 1888 it ranged between $37 and $48.

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